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ITS

STATISTICS, TECHNOLOGY AND TRADE

DURING

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FOUNDED BY RICHARD P. ROTHWELL

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WALTER RENTON INGALLS
Editor of the Engineering and Mining Journal; Member American Institute of Mining Engineers; Member Institution of Mining and Metallurgy; Member American Chemical Society; Member Society of Chemical Industry, Etc.

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CONTRIBUTORS TO VOLUME XIV.

Aubrey, A. J.
Chemist, Laclede Firebrick Company, St. Louis, Mo.
The Refractory Uses of Bauxite.

Austin, L. S.
Professor of Metallurgy, State School of Mines, Houghton, Mich.
Progress in the Metallurgy of Copper in 1905.

Bell, Robert N.
State Mine Inspector, Idaho.
Lead in Idaho.
Gold and Silver in Idaho.

Brinsmade, Robert B.
Professor of Mining, New Mexico State School of Mines, Socorro, New Mexico.
Pyrites Mining and Milling, St. Lawrence County, N. Y.
Talc in Northern New York.

Brown, A. Selwyn-.
Consulting Mining Engineer. New York City.
Gold Mining in Nevada.

Collins, George E.
Consulting Mining Engineer, Denver, Colo.
Gold and Silver Mining in Colorado.

Cushing, George H.
Cleveland, Ohio.
The Cleveland Iron Market.

Dorr, J. V. N.
Consulting Engineer and General Manager, Buxton & Big Bonanza Mines, Lead, South Dakota.
Gold Mining in South Dakota.

Douglas, James.
President Copper Queen Consolidated Mining Co., New York City.
Copper in Arizona.

Easton, Stanley A.
General Manager, Bunker Hill and Sullivan Mining Co., Kellogg, Idaho.
Lead Mining in the Cœur d'Alene.

Fohs, F. Julius
Assistant Geologist, Kentucky Geological Survey, Louisville, Ky.
Fluorspar.

Friedman, L. W.
Birmingham, Ala.
The Alabama Iron Market.

Fulton, Charles H.
President, South Dakota School of Mines, Rapid City, South Dakota.
Cyanidation during 1905.

George, Harold C.
Instructor, School of Mines and Mining, Western University of Pennsylvania, Allegheny City, Pa.
Pennsylvania Oil Fields.

Haworth, Erasmus.
State Geologist and Professor of Geology, University of Kansas, Lawrence, Kan.
Oil and Gas Development in the Mid-Continental Field.

Hobart, Frederick.
Associate Editor, *Engineering and Mining Journal.*
Coal and Coke.
Gold and Silver.
Iron and Steel.

Hofman, H. O.
Professor of Metallurgy, Massachusetts Institute of Technology, Boston, Mass.
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Hoover, H. C.
Consulting Mining Engineer, London, England.
Gold Mining in Western Australia.

Hutchins, J. P.
Consulting Mining Engineer, New York City.
Gold Dredging in 1905.
Mining in the Yukon.

Ingalls, W. R.
Editor of *Engineering and Mining Journal.*
Copper.
Lead.
Tin.
Zinc.

Jacobs, E.
Editor of *British Columbia Mining Record*, Victoria, B. C.
Gold Mining in British Columbia.

James, Alfred.
Vice-President Institution of Mining and Metallurgy and Consulting Engineer, London, England.
Progress in Gold Ore Treatment during 1905.

Johnson, R. D. O.
Consulting Mining Engineer, Denver, Colorado.
Phosphate Rock in Tennessee.

Judd, Edward K.
Editorial Staff, *Engineering and Mining Journal.*
Aluminum.
Antimony.
Chromium and Chrome Ore.
Corundum and Emery.
Graphite.
Nickel and Cobalt.
Sulphur and Pyrites.
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Keyes, Chas. R.
President, New Mexico State School of Mines, Socorro, New Mexico.
Gold Mining in New Mexico.
Copper Mining in New Mexico.

Lesley, Robert W.
Vice-President, Society Testing Materials; Vice-President, American Cement Company; Past President, Association American Portland Cement Manufacturers; Philadelphia, Pa.
Cement.

LUCAS, CAPT. J. B.
Mining Engineer, Washington, D. C.
Coal and Coke in Montana.

LUTY, S. F.
Pittsburgh, Pa.
The Pittsburgh Iron and Steel Markets.

MALCOLMSON, JAMES W.
Consulting Mining Engineer, El Paso, Texas.
Gold and Silver Mining in Mexico.

MANCE, F. S.
Department of Mines, Sydney, New South Wales.
Gold Mining in New South Wales.
Tin Mining in New South Wales.

MARSTERS, V. F.
Geologist of the Republic of Peru, Lima, Peru.
Petroleum in Peru.

MEEKS, REGINALD.
Editorial Staff, *Engineering and Mining Journal.*
Potassium Salts.
Sodium and Sodium Salts.
Tungsten.

MINCHIN, JOHN B.
Mining Engineer, Oruro, Bolivia.
Tin in Bolivia.

MOORE, E. W.
General Manager, Galena Iron Works, Platteville, Wis.
Zinc Mining in Wisconsin.

MORRISON, E.
Chicago, Ill.
The Chicago Iron and Steel Markets.

NEWLAND, D. H.
Assistant State Geologist, Albany, New York.
Garnet.
Graphite in New York.

PAINTER, ROBERT K.
Formerly Superintendent, Cabin Branch Mine, Virginia.
Pyrites Mining and Milling in Virginia.

PARSONS, F. W.
Associate Editor, *Engineering and Mining Journal.*
Petroleum.

PAUL, JAMES W.
Chief Mine Inspector, Charleston, W. Va.
Coal and Coke in West Virginia.

PHILLIPS, WILLIAM B.
General Manager Chisos Mining Co., Terlingua, Texas.
The Quicksilver Industry in Brewster County, Texas.

RICE, CLAUDE T.
Editorial Staff, *Engineering and Mining Journal.*
The Emery Industry at Chester, Mass.

RICHARDS, ROBERT H.
Professor of Mining, Massachusetts Institute of Technology, Boston, Mass.
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SPEIER, PAUL.
Metal Merchant, Breslau, Germany.
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STOUGHTON, BRADLEY.
Adjunct Professor of Metallurgy, Columbia School of Mines, New York City.
Progress of the Metallurgy of Iron and Steel in 1905.

THOMAS, C. E. L.
Calumet, Mich.
Copper in Michigan.

VAN MATER, J. A.
General Manager, Bertha Mineral Co., Pulaski, Virginia.
Lead and Zinc Mining in Virginia.

WHEELER, H. A.
Consulting Mining Engineer, St. Louis, Mo.
Lead in Southeastern Missouri.

WILKINSON, W. FISCHER.
Engineering Staff, Consolidated Gold Fields of South Africa, Ltd., London, England.
Gold Mining in the Transvaal.
Gold Mining in Rhodesia.

WOODBRIDGE, DWIGHT E.
Consulting Mining Engineer, Duluth, Minn.
Lake Superior Iron Mines.

YALE, CHAS. G.
Associate Editor, *Engineering and Mining Journal*, San Francisco, Cal.
Gold Mining in California.
Quicksilver in California.

ZOOK, JESSE A.
Joplin, Mo.
Lead in Southwestern Missouri.
Zinc in Missouri and Kansas.

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25·4000	0·304801	0·914402	1·60935	= 1 =	3·70	29·57	0·94636	3·78544	16·387	0·02832	0·765	0·35242
50·8001	0·609601	1·828804	3·21869	= 2 =	7·39	59·15	1·89272	7·57088	32·774	0·05663	1·529	0·70485
76·2001	0·914402	2·743205	4·82804	= 3 =	11·09	88·72	2·83908	11·35632	49·161	0·08495	2·294	1·05727
101·6002	1·219202	3·657607	6·43739	= 4 =	14·79	118·30	3·78544	15·14176	65·549	0·11327	3·058	1·40969
127·0002	1·524003	4·572009	8·04674	= 5 =	18·48	147·87	4·73180	18·92720	81·936	0·14158	3·823	1·76211
152·4003	1·828804	5·486411	9·65608	= 6 =	22·18	177·44	5·67816	22·71264	98·323	0·16990	4·587	2·11454
177·8003	2·133604	6·400813	11·26543	= 7 =	25·88	207·02	6·62452	26·49808	114·710	0·19822	5·352	2·46696
203·2004	2·438405	7·315215	12·87478	= 8 =	29·57	236·59	7·57088	30·28352	131·097	0·22654	6·116	2·81938
228·6004	2·743205	8·229616	14·48412	= 9 =	33·28	266·16	8·51724	34·06896	147·484	0·25485	6·881	3·17181

Square.					Weight.			
Square Inches to Square Centimeters.	Square Feet to Square Decimeters.	Square Yards to Square Meters.	Acres to Hectares.		Grains to Milligrams.	Avoirdupois Ounces to Grams.	Avoirdupois Pounds to Kilograms.	Troy Ounces to Grams.
6·452	9·290	0·836	0·4047	=1=	64·7989	28·3495	0·45359	31·10348
12·503	18·581	1·672	0·8094	=2=	129·5978	56·6991	0·90719	62·20696
19·355	27·871	2·508	1·2141	=3=	194·3968	85·0486	1·36078	93·31044
25·807	37·161	3·344	1·6187	=4=	259·1957	113·3981	1·81437	124·41392
32·258	46·452	4·181	2·0234	=5=	323·9946	141·7476	2·26796	155·51740
38·710	55·742	5·017	2·4281	=6=	388·7935	170·0972	2·72156	186·62089
45·161	65·032	5·853	2·8328	=7=	453·5924	198·4467	3·17515	217·72437
51·613	74·323	6·689	3·2375	=8=	518·3914	226·7962	3·62874	248·82785
58·065	83·613	7·525	3·6422	=9=	583·1903	255·1457	4·08233	279·93133

1 chain = 20·1160 meters.
1 square mile = 259 hectares.
1 fathom = 1·829 meters.
1 nautical mile = 1853·27 meters.
1 foot = 0·304801 meter, 9·4840158 log.
1 avoir. pound = 453·5924277 gram.
15432·35639 grains = 1 kilogram.

TABLES FOR CONVERTING METRIC TO UNITED STATES WEIGHTS AND MEASURES.

Linear.					Capacity.							
Meters to Inches.	Meters to Feet.	Meters to Yards.	Kilometers to Miles.		Milliliters or Cubic Centiliters to Fluid Drams.	Centiliters to Fluid Ounces.	Liters to Quarts.	Dekaliters to Gallons.	Hectoliters to Bushels.	Cubic Centimeters to Cubic Inches.	Cubic Meters to Cubic Feet.	Cubic Meters to Cubic Yards.
39·3700	3·28083	1·093611	0·62137	= 1 =	0·27	0·338	1·0567	2·6417	2·8375	0·0610	35·314	1·308
78·7400	6·56167	2·187222	1·24274	= 2 =	0·54	0·676	2·1134	5·2834	5·6750	0·1220	70·629	2·616
118·1100	9·84250	3·280833	1·86411	= 3 =	0·81	1·014	3·1700	7·9251	8·5125	0·1831	105·943	3·924
157·4800	13·12333	4·374444	2·48548	= 4 =	1·08	1·352	4·2267	10·5668	11·3500	0·2441	141·258	5·232
196·8500	16·40117	5·468056	3·10685	= 5 =	1·35	1·691	5·2834	13·2085	14·1875	0·3051	176·572	6·540
236·2200	19·68500	6·561667	3·72822	= 6 =	1·62	2·029	6·3401	15·8502	17·0250	0·3661	211·887	7·848
275·5900	22·96583	7·655278	4·34959	= 7 =	1·89	2·368	7·3968	18·4919	19·8625	0·4272	247·201	9·156
314·9600	26·24667	8·748889	4·97096	= 8 =	2·16	2·706	8·4534	21·1336	22·7000	0·4882	282·516	10·464
354·3300	29·52750	9·842500	5·59233	= 9 =	2·43	3·043	9·5101	23·7753	25·5375	0·5492	317·830	11·771

Square.					Weight.			
Square Centimeters to Square Inches.	Square Meters to Square Feet.	Square Meters to Square Yards.	Hectares to Acres.		Kilograms to Grains.	Hectograms to Ounces Avoirdupois	Kilograms to Pounds Avoirdupois.	Grams to Ounces Troy.
0·1550	10·764	1·196	2·471	= 1 =	15432·36	3·5274	2·20462	0·03215
0·3100	21·528	2·392	4·942	= 2 =	30864·71	7·0548	4·40924	0·06430
0·4650	32·292	3·588	7·413	= 3 =	46297·07	10·5822	6·61386	0·09645
0·6200	43·055	4·784	9·884	= 4 =	61729·43	14·1096	8·81849	0·12860
0·7750	53·819	5·980	12·355	= 5 =	77161·78	17·6370	11·02311	0·16075
0·9350	64·583	7·176	14·826	= 6 =	92594·14	21·1644	13·22773	0·19290
1·0850	75·347	8·372	17·297	= 7 =	108026·49	24·6918	15·43235	0·22505
1·2400	86·111	9·568	19·768	= 8 =	123458·85	28·2192	17·63697	0·25721
1·3950	96·874	10·764	22·2	= 9 =	138891·21	31·7466	19·84159	0·28936

The only material standard of customary length authorized by the U. S. Government is the Troughton scale, whose length at 59.62 Fahr. conforms to the British standard. The yard in use in the United States is therefore equal to the British yard.

The only authorized material standard of customary weight is the Troy pound (5,760 grains) of the Mint. It is of brass of unknown density, and therefore not suitable for a standard of mass. It was derived from the British standard Troy pound of 1758 by direct comparison. The British avoirdupois pound was also derived from the latter, and contains 7,000 grains troy.

The grain Troy is therefore the same as the grain avoirdupois, and the pound avoirdupois in use in the United States is equal to the British pound avoirdupois.

The British gallon = 4·54346 liters.

The British bushel = 36·3477 liters.

By the concurrent action of the principal Governments of the world an International Bureau of Weights and Measures has been established near Paris. Under the direction of the International Committee, two ingots were cast of pure platinum-iridium in the proportion of 9 parts of the former to 1 of the latter metal. From one of these a certain number of kilograms were prepared, from the other a definite number of meter bars. These standards of weight and length were intercompared, without preference, and certain ones were selected as International prototype standards. The others were distributed by lot to the different Governments and are called National prototype standards.

The metric system was legalized in the United States in 1866.

The International Standard Meter is derived from the Metre des Archives, and its length is defined by the distance between two lines at 0° Centigrade, on a platinum-iridium bar deposited at the International Bureau of Weights and Measures.

The International Standard Kilogram is a mass of platinum-iridium deposited at the same place, and its weight *in vacuo* is the same as that of the Kilogramme des Archives.

The liter is equal to a cubic decimeter of water, and it is measured by the quantity of distilled water which, at its maximum density, will counterpoise the standard kilogram in a vacuum, the volume of such a quantity of water being, as nearly as has been ascertained, equal to a cubic decimeter.

Long ton: 2240 lb. avoirdupois	=	1016	kilogram.
Short ton: 2000 " "	=	907·2	"
Pound avoirdupois	=	453·6	grams.
Flask of Mercury = 76·5 lb. avoir.	=	34·7	kilograms.
Troy ounce	=	31·104	grams.
Gallon	=	2·785	liters.

Barrel of petroleum	= 42 gal.	=	1·59	hectoliter.
" " salt	= 280 lb.	=	127	kilogram.
" " lime	= 200 "	=	90·720	"
" " natural cement	= 300 "	=	136·080	"
" " Portland cement	= 400 "	=	181·440	"

Gold coining value per oz. Troy $20·6718 = $0·6646 per gram.
Silver " " " Troy $1·2929 = $0·04157 "

VALUES OF FOREIGN COINS

COUNTRY	Standard	Monetary Unit	Value in Terms of U. S. Gold Dollar	Coins
Argentine Republic	Gold	Peso	$0.965	Gold: argentine ($4.824) and ½ argentine Silver: peso and divisions.
Austria-Hungary	Gold	Crown	.203	Gold: 10 and 20 crowns. Silver: 1 and 5 crowns
Belgium	Gold	Franc	.193	Gold: 10 and 20 francs. Silver: 5 francs.
Bolivia	Silver	Boliviano	.422	Silver: boliviano and divisions.
Brazil	Gold	Milreis	.546	Gold: 5, 10 and 20 milreis. Silver: ½, 1, and 2 mil reis.
British Possessions, N. A. (except Newf'nd).	Gold	Dollar	1.000	
Central Amer. States— Costa Rica	Gold	Colon	.465	Gold: 2, 5, 10, and 20 colons ($9.307). Silver: 5, 10 25, and 50 centimos.
British Honduras	Gold	Dollar	1.000	
Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	.422	Silver: peso and divisions.
Chile	Gold	Peso	.365	Gold: escudo ($1.825), doubloon ($3.650), and condor ($7.300). Silver: peso and divisions.
China	Silver	Tael: Amoy	.691	
		Tael: Canton	.689	
		Tael: Chefoo	.661	
		Tael: Chin Kiang	.675	
		Tael: Fuchau	.639	
		Tael: Haikwan (Customs)	.703	
		Tael: Hankow	.647	
		Tael: Kiaochow	.682	
		Tael: Nankin	.684	
		Tael: Niuchwang	.648	
		Tael: Ningpo	.664	
		Tael: Pekin	.674	
		Tael: Shanghai	.631	
		Tael: Swatow	.638	
		Tael: Takau	.695	
		Tael: Tientsin	.670	
		Dollar: Hongkong	.455	
		Dollar: British	.455	
		Dollar: Mexican Chopped	.458	
Colombia	Gold	Dollar	1.000	Gold: condor ($9.647) and double condor. Silver: peso.
Denmark	Gold	Crown	.268	Gold: 10 and 20 crowns.
Ecuador	Gold	Sucre	.487	Gold: 10 sucres ($4.8665). Silver: sucre and divisions.
Egypt	Gold	Pound (100 piastres)	4.943	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters Silver: 1, 2, 5, 10, and 20 piasters.
Finland	Gold	Mark	.193	Gold: 20 marks ($3.859), 10 marks ($1.93).
France	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
German Empire	Gold	Mark	.238	Gold: 5, 10 and 20 marks.
Great Britain	Gold	Pound sterling	4.866½	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Gold	Drachma	.193	Gold: 5, 10, 20, 50, and 100 drachmas. Silver: 5 drachmas.
Haiti	Gold	Gourde	.965	Gold: 1, 2, 5, and 10 gourdes. Silver: gourde and divisions.
India	Gold	Pound sterling*	4.866½	Gold: sovereign (pound sterling). Silver: rupee and divisions.
Italy	Gold	Lira	.193	Gold: 5, 10, 20, 50, and 100 lira. Silver: 5 lira.

NOTE.—The coins of silver-standard countries are valued by their pure silver contents, at the average market price of silver for the three months preceding July 1, 1905.

*The sovereign is the standard coin of India, but the rupee ($0.3244⅓) is the money of account, current at 15 to the sovereign.

COUNTRY	Standard	Monetary Unit	Value in Terms of U. S. Gold Dollar	Coins
Japan	Gold	Yen	.498	Gold: 5, 10, and 20 yen. Silver: 10, 20, and 50 sen.
Liberia	Gold	Dollar	1.000	
Mexico	Gold	Peso†	.498	Gold: 5 and 10 pesos. Silver: dollar‡ (or peso) and divisions.
Netherlands	Gold	Florin	.402	Gold: 10 florins. Silver: ½, 1, and 2½ florins.
Newfoundland	Gold	Dollar	1.014	Gold: 2 dollars ($2.027).
Norway	Gold	Crown	.268	Gold: 10 and 20 crowns.
Panama	Gold	Balboa	1.000	Gold: 1, 2½, 5, 10, and 20 balboas. Silver: peso and divisions.
Persia	Silver	Kran	.078	Gold: ⅕, 1 and 2 tomans ($3.409). Silver: ¼, ½, 1, 2, and 5 krans.
Peru	Gold	Sol	.487	Gold: libra ($4.8665). Silver: sol and divisions.
Philippine Islands	Gold	Peso	.500	Silver: peso, 10, 20, and 50 centavos.
Portugal	Gold	Milreis	1.080	Gold: 1, 2, 5, and 10 milreis.
Russia	Gold	Ruble	.515	Gold: 5, 7½, 10, and 15 rubles. Silver: 5, 10, 15, 20, 25, 50, and 100 copecks.
Spain	Gold	Peseta	.193	Gold: 25 pesetas. Silver: 5 pesetas.
Sweden	Gold	Crown	.268	Gold: 10 and 20 crowns.
Switzerland	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	.044	Gold: 25, 50, 100, 250, and 500 piasters.
Uruguay	Gold	Peso	1.034	Gold: peso. Silver: peso and divisions.
Venezuela	Gold	Bolivar	.193	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

† Seventy-five centigrams fine gold.
‡ Value in Mexico, $0.498.

INTRODUCTION.

In the preparation of the statistics for this volume, the figures previously reported for 1904 have been revised in the light of later and more minute investigation, in accordance with our practice, therefore it is important for students to observe the caution to use always the figures in the latest volume of The Mineral Industry. There are no statistical reports of this nature which are absolutely correct, owing to the practical impossibility of obtaining accurate reports from all the producers in some extensive and greatly subdivided industries, the absence of records on the part of many producers, which prevents them from making returns, the unwillingness of a few to give their figures, and the confusion as to the stage in which many products are to be reported. The last difficulty is especially likely to lead to errors in values, some producers estimating the worth of their product at the pit's mouth, and others reporting it in a more or less advanced state of completion, including thus not only the cost of carriage, but also the cost of manipulation. These difficulties appear not only in our own statistics, but also in the statistics reported by various governments. In our own work, however, we make a practice of going backward and correcting figures previously reported, whenever mistakes are discovered by subsequent investigation. In estimating values, we are disposed, therefore, to use actual market prices rather than the values reported by the producers themselves, which are apt to be misleading for the reasons mentioned above.

For many of the statistics relating to the mineral production of the United States in 1905 and previous years, we are indebted to the United States Geological Survey, and for the production of gold and silver in the United States to George E. Roberts, Director of the Mint. Acknowledgment is due, also, to various State geological surveys and statistical bureaus for information incorporated in this volume. We have generally credited such information to the proper sources, but this acknowledgment may stand for any unintentional oversights. The same acknowledgment is due with respect to the foreign statistics, which we state always as officially reported by the respective governments, when such reports are available.

In restoring this volume of The Mineral Industry to the original promptness of publication, the editorial work upon it having been completed at the end of May, it has been impossible to collect statistics for

all the substances of mineral production in the United States, but the omissions are generally in the cases of substances of minor importance.

Moreover some of the statistics reported in this volume are preliminary, and subject to revision. It is our belief, however, that statistics of reasonable commercial accuracy, promptly published, are of greater value to technology and trade than are statistics, corrected to the last unit, which are published a year or two late. However, it has been necessary to use these approximations in only a few instances, and most of the figures which are to be found in this book will prove to require only slight, if any, revision.

We have divided the large table, which previously followed this introduction, into three sections, one representing the production of ores and minerals in the United States; the second representing the production of metals of domestic origin; and the third representing the manufacture of chemical products, or other articles of commerce that are derived from the ores or minerals directly mined. It is conceived that this arrangement is more logical, and more valuable than one in which all the substances are heterogeneously assembled. Values are reported merely as an indication of the relative magnitude of the various industries, from the commercial standpoint. As previously noted, comparatively little weight is to be placed upon value figures. For example, as a matter of form, all the copper produced in the United States is computed on the basis of the average price for Lake copper, in conformity with previously existing custom, although, as it appears in the detailed statistics, by far the major portion of the copper produced in the United States is sold as electrolytic copper, which fetches a slightly lower price than Lake; there are similar differences with regard to spelter, and other metals.

Ores and Minerals.

Asbestos.—The output of asbestos in 1905 was 3100 short tons, valued at $126,300, or nearly double the quantity produced in 1904. The high valuation was due to the larger amount of chrysotile asbestos included in the output.

Asphalt.—The production of asphaltic materials in 1905 was as follows, the corresponding quantities and values for 1904 being given in parentheses for comparison: Asphaltum, including the mineral waxes, 185,-703 short tons, valued at $1,567,397 (85,130 tons—$1,083,721); asphaltic limestone, 2000 tons, valued at $5000 (1798 tons—$4495); bituminous sandstone, 36,530 short tons, valued at $116,885 (19,641 tons—$71,465). The most noteworthy progress in the industry occurred in Texas, whose output of solid and liquid asphalt showed a remarkable increase.

Barytes.—The production of barytes amounted to 53,252 short tons,

valued at $196,041, as compared with 65,727 tons, worth $174,958, in the preceding year. Missouri is the largest single producer of barytes.

Bauxite.—The production of bauxite in 1905 is estimated at 47,991 long tons, valued at $203,960, as compared with 48,012 tons, worth $166,121, in 1904. The bauxite resources of the United States are gradually coming under the control of the aluminum-producing interests which now control most of the output of Arkansas and a large part of that of the South.

Diatomaceous Earth.—The production of diatomaceous earth in 1905 amounted to 10,977 short tons, valued at $64,637, as compared with 6274 tons, worth $44,164, the output of 1904. Only a small part of the output of this material is used as abrasive, the principal consumption being in the manufacture of dynamite. Missouri is the largest contributing State.

Emery.—The output of emery in the United States amounted to 2315 short tons, valued at $19,677, as compared with 1932 tons, worth $57,235, in 1904. The output comes from Peekskill, N. Y., with a little from Chester, Mass.

Fluorspar.—The production of fluorspar in 1905 amounted to 50,340 short tons, valued at $295,496, as compared with 36,452 tons, worth $234,755, in the preceding year. Practically the entire output is mined within a small area lying around the junction of the Ohio with the Mississippi River.

Fuller's Earth.—The production of fuller's earth in 1905 amounted to 21,745 short tons, valued at $157,776, as against 29,480 tons, worth $168,500, in the previous year. Florida is the largest producing State in the Union, where most of the product used for clarifying mineral and vegetable oils is obtained.

Garnet.—The output of garnet in the United States was 3694 short tons, valued at $114,625, as compared with 2952 tons, worth $89,636, in the preceding year. New York is the largest producing State. The annual production of garnet does not seem to suffer by the rapid advance in the use of artificial abrasives.

Graphite.—The production of crystalline graphite in the United States in 1905 amounted to 4,260,656 ℔., estimated to have a value of $170,426, as compared with 4,357,927 ℔., worth $162,332, in the previous year. Much the largest part of this output comes from the Adirondack mountains region of New York, with a little contributed by Chester county, Pa. The United States product of crystalline graphite is almost uniformly of the flaky variety, and it is therefore better suited to the manufacture of pencils and lubricants than to refractory ware. The annual production of crystalline graphite in the United States has ranged closely around 4,000,000 ℔. for the last seven years.

Iron Ore.—The output of iron ore in the United States in 1905 reached

the total of 44,578,456 long tons, the value of which is estimated at $94,-768,122, as compared with 29,462,839 long tons in 1904, valued at $51,559,-968. This output, the largest yet recorded, was contributed by the principal producing districts in the following order: Lake Superior district, 34,353,456 long tons; the Southern States, 7,175,000; other States, principally New York and the Central States, 3,050,000 tons.

Limestone Flux.—The amount of limestone used as a flux in iron, lead and copper smelting in 1905 was 14,098,000 long tons, which cost $6,739,-200 for quarrying, as against 10,657,038 tons, costing $4,702,768, in the previous year. The iron industry naturally consumed the largest part of this material.

Molybdenum Ore.—The output of molybdenum ore concentrate in 1905 was only 6 short tons, with a value of $1,050, as compared with 15 tons, worth $2175, in 1904. Practically the entire output comes from the

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES.

Products.	Measures.	1904.		1905.	
		Quantity.	Value.	Quantity.	Value.
Asbestos	Sh. T.	1,480	$25,740	3,100	$126,300
Asphaltum	Sh. T.	85,130	1,083,721	185,703	1,567,397
Asphaltic limestone	Sh. T.	1,798	4,495	2,000	5,000
Bituminous sandstone	Sh. T.	19,641	71,465	36,530	116,885
Barytes	Sh. T.	65,727	174,958	53,252	196,041
Bauxite	L. T.	48,012	166,121	47,991	203,960
Calcium borate, crude	Sh. T.	45,647	698,810	....	....
Chrome ore	L. T.	123	1,845	(*e*) 150	2,250
Coal, anthracite	Sh. T.	73,674,480	162,151,898	78,731,523	178,788,244
Coal, bituminous	Sh. T.	277,512,729	326,482,886	310,040,644	354,011,988
Diatomaceous earth	Sh. T.	6,274	44,164	10,977	64,637
Emery	Sh. T.	1,932	57,235	2,315	19,677
Feldspar	Sh. T.	45,188	266,326	35,419	226,157
Flint	Sh. T.	52,270	100,590	51,145	104,109
Fluorspar	Sh. T.	36,452	234,755	50,340	295,496
Fuller's earth	Sh. T.	29,480	168,500	21,745	157,776
Garnet	Sh. T.	2,952	89,636	3,694	114,625
Graphite, amorphous	Sh. T.	19,115	102,925	(*e*) 10,000	50,000
Graphite, crystalline	℔	4,357,927	162,332	4,260,656	170,426
Grindstones	......	....	881,527	....	777,606
Gypsum	Sh. T.	940,917	2,784,325	....	....
Iron ore	L. T.	29,462,839	51,559,968	44,578,456	94,768,122
Limestone flux	L. T.	10,657,038	4,703,768	14,098,000	6,739,200
Magnesite	Sh. T.	2,850	9,298	(*e*) 3,000	10,500
Manganese ore (*d*)	L. T.	454,581	789,132	....	....
Mica, sheet	℔	668,358	109,462	(*e*) 700,000	110,000
Mica, scrap	Sh. T.	1,096	10,854	(*e*) 1,200	12,000
Millstones	......	....	37,338	....	37,974
Molybdenum ore	Sh. T.	15	2,175	6	1,050
Monazite	℔	745,999	85,038	1,352,418	163,908
Ocher (*j*)	Sh. T.	24,797	261,299	(*e*) 25,000	270,000
Petroleum, crude	Bbl. (*i*)	118,396,335	102,294,433	139,889,210	118,905,828
Phosphate rock	L. T.	1,874,428	6,873,625	1,933,286	9,713,296
Precious stones	......	....	315,900	....	326,350
Pumice	Sh. T.	1,530	5,421	1,832	5,540
Pyrites	L. T.	173,221	669,124	200,280	651,796
Quartz, crystalline	Sh. T.	31,924	74,600	19,039	88,118
Salt (*k*)	Bbl.	22,030,002	6,021,222	25,966,122	6,095,922
Sand, glass	Sh. T.	858,719	796,492	(*e*) 900,000	810,000
Slate, roofing	Squares (*f*)	1,233,757	4,669,289	1,241,227	4,574,550
Soda, natural	Sh. T.	12,000	18,000	(*e*) 12,000	18,000
Sulphur	L. T.	193,492	3,869,840	232,000	4,872,000
Talc, common	Sh. T.	27,184	433,331	40,134	637,062
Talc, fibrous	Sh. T.	64,005	507,400	67,000	469,000
Tungsten ore	Sh. T.	740	184,000	834	257,493
Whetstones and oilstones	......	....	188,985	....	244,346
Zinc ore	Sh. T.	693,025	12,071,456	795,698	15,596,457
Total enumerated	......	....	$692,315,704	....	$802,377,086

PRODUCTION OF SECONDARY MINERALS AND CHEMICALS IN THE UNITED STATES.

Products.	Measures.	1904.		1905.	
		Quantity.	Value.	Quantity.	Value.
Alundum	℔	4,020,000	$281,400	3,612,000	$252,840
Arsenic	Sh. T.	498	29,504	773	50,225
Borax	Sh. T.	....	....	....	....
Bromine	℔	879,312	215,431	899,434	139,432
Carborundum	℔	7,060,380	706,038	5,596,280	599,628
Cement, nat. hyd	Bbl (*g*)	4,866,331	2,450,150	4,473,049	2,413,052
Cement, portland	Bbl (*h*)	26,505,881	23,355,119	36,033,012	33,326,523
Cement, slag	Bbl (*h*)	303,045	226,651	382,447	272,614
Cobalt oxide	℔	22,000	42,600	....	....
Coke	Sh. T.	22,005,561	54,418,200	28,404,112	72,284,336
Copper sulphate (*c*)	℔	63,234,557	3,161,728	52,278,996	2,352,555
Copperas	Sh. T.	16,956	118,692	21,103	147,721
Crushed steel	℔	790,000	55,300	812,000	56,840
Graphite, artificial	℔	3,248,000	217,790	4,595,500	313,979
Lead, white	Sh. T.	126,336	13,896,913	122,398	12,068,443
Lead, sublimed white	Sh. T.	6,478	647,800	6,977	697,700
Lead, red	Sh. T.	13,938	1,672,569	16,269	1,919,767
Lead, orange mineral	Sh. T.	1,125	168,681	1,000	120,000
Litharge	Sh. T.	12,487	1,248,691	12,643	1,422,616
Zinc white (*m*)	Sh. T.	59,613	4,524,031	65,403	5,232,240
Zinc-lead	Sh. T.	6,781	474,670	7,200	540,000
Total	.....	....	$107,911,958	....	$134,250,511

PRODUCTION OF METALS IN THE UNITED STATES

Products.	Measures.	1904.		1905.	
		Quantity.	Value.	Quantity.	Value.
Aluminum	℔	8,600,000	$2,477,000	10,000,000	$3,200,000
Antimony	℔	5,854,000	372,958	5,912,000	614,848
Copper	℔	817,715,005	106,221,179	871,634,245	136,837,860
Ferromanganese (*q*)	L. T.	219,446	9,304,510	289,983	17,639,666
Gold (fine)	Troy oz.	3,904,986	80,723,200	4,260,504	87,948,237
Iron, pig	L. T.	16,277,587	225,281,804	22,702,397	377,540,862
Lead	Sh. T.	302,204	26,043,941	322,474	30,357,702
Nickel	℔	24,000	11,400	(*s*)	....
Platinum	Troy oz.	200	2,600	200	4,000
Quicksilver	Flasks (*o*)	35,244	1,489,716	30,650	1,189,220
Silver (fine)	Troy oz.	57,786,100	33,515,938	58,918,839	35,850,955
Zinc	Sh. T	181,803	18,543,906	201,748	23,733,635
Total metals	....	....	$503,988,152	....	$714,916,985
Total ores and minerals	....	....	692,315,704	....	802,377,086
Secondary products	....	....	107,911,958	....	134,250,511
Grand Total	....	....	1,304,215,814	....	1,651,544,582

Additional details will be found under the respective captions farther on in this volume. (*c*) Includes sulphate made from metallic copper. (*d*) Includes manganiferous iron ore. (*e*) Estimated. (*f*) One "square" covers 100 square feet. (*g*) Barrels of 300 lb. (*h*) Barrels of 380 lb. (*i*) Barrels of 42 gallons. (*j*) Includes ocher, umber, sienna and Venetian red. (*k*) Includes salt used in manufacture of alkali; the barrel of salt weighs 280 lb. (*m*) Includes a small quantity made from spelter. (*o*) Flasks of 75 lb. (*q*) Includes spiegeleisen, although the value is given as for ferromanganese. (*s*) Insignificant.

Troy district, Pinal county, Ariz., where it is recovered as a by-product in the treatment of copper ores.

Monazite.—The output of monazite, including a very small production of certain other rare minerals, in the United States in 1905 amounted to 1,352,418 ℔., with a value of $163,908, as compared with 745,999 ℔., worth $85,038, in 1904. The entire production of monazite comes from the Carolinas, although a possible new source of the mineral has been discovered in the black sand from certain localities in Washington and Oregon, which were collected and carefully studied at Portland, Ore., during 1905.

Phosphate Rock.—We are unable to give exact statistics of production of this material during 1905; shipments to foreign and domestic points by rail and by water in 1905 amounted to 1,933,286 long tons, with an estimated value of $9,713,296, as compared with production amounting to 1,874,428 tons, worth $6,873,625, in 1904. Florida contributes much the larger part of this output and the only other producing States are Tennessee and South Carolina. Most of the Florida rock is exported, but most of that mined in Tennessee and South Carolina is consumed by fertilizer manufacturers close to the points where it is mined.

Precious Stones.—The value of the output of precious stones of all kinds recovered in the United States in 1905 was $326,350, as compared with $324,300, the value of the stones recovered in 1904. Of this output, sapphires constitute nearly half the value, and are followed in importance by turquoise and tourmaline. The United States produces a large variety of semi-precious stones, which have been cut and set for jewelry, but as a producer of gems of first quality the United States is not important. In 1905 this country produced neither diamonds, rubies, nor emeralds. Its output of sapphires, valued at $125,000, is an important item in the world's production, and the stones mined at Yogo gulch, Mont., are second in quality to no others.

Pyrites.—The output of pyrites in the United States in 1905 was 200,280 long tons, valued at $651,796, as compared with 173,221 tons, worth $669,124, in the preceding year. Virginia contributes about half the entire output, and is followed by Massachusetts, New York, California and Ohio.

Crystalline Quartz.—The output of quartz used for abrasive purposes in the United States in 1905, including also a small output of feldspar used in the same way, was 19,039 short tons, valued at $88,718, as compared with 31,940 tons, worth $74,850, in 1904. This does not include large quantities of sand used by stone cutters and other trades, for which statistics are not available. The quartz enumerated in the above total comes from New York, Connecticut, Pennsylvania, and Wisconsin, and is used in the manufacture of sandpaper and scouring soaps, and as a wood filler. It is important to note that the above output includes only that material used as abrasive; the output of quartz glass sands and of feldspar used by the pottery trade is enumerated in other paragraphs.

Sulphur.—It is impossible to give exact statistics of the output of sulphur in 1905. We estimate it, from a consideration of consumption and imports, to have been 232,000 long tons. On the basis of the average quotation at New York for the year, this output had a value of $4,852,000. The output in 1904, estimated on the same basis, was 193,492 long tons, valued at $3,869,840. Practically the entire output comes from the recently opened deposits of Louisiana.

Talc and Soapstone.—The total production of talc and soapstone in the

United States in 1905 amounted to 107,134 short tons, valued at $1,106,-062, as compared with 81,189 tons, valued at $940,731, in the preceding year. The output in 1905 was constituted by 67,000 tons of fibrous talc, all of which was mined in New York, and 40,134 tons of ordinary talc and soapstone, which was contributed by New England and Southern States. The New York product is all used in the paper trade, and that from other sections of the country is mainly sawed into slabs and manufactured into various kinds of articles.

Tungsten Ore.—The output of tungsten ore concentrate in 1905 amounted to 834 short tons, valued at $257,463, as compared with 740 tons, worth $184,000, in 1904. We are unable to state the amount of crude ore that went into the making of this concentrate, but the average tungsten content of the larger part of the above total was above 60 per cent. Colorado is the principal contributor of tungsten ore; a number of concentrating mills are located in Boulder county of this State, whose principal output is tungsten concentrate. California, Idaho and Arizona contribute most of the remainder.

Zinc Ore.—The output of zinc ore in 1905 was 795,698 short tons, valued at $15,596,467, as compared with an output of 693,025 short tons, valued at $12,071,456, in the preceding year. For details of this production we refer to the article on a following page.

Secondary Mineral and Chemical Products.

Alundum.—The production of alundum, or artificial corundum (aluminum oxide), in 1905 amounted to 3,612,000 lb., valued at $252,840, as compared with an output of 4,020,000 lb. in 1904. The entire output is made by the Norton Emery Wheel Company, of Worcester, Mass., and is used by this company in the manufacture of its corundum wheels. The product is made at Niagara Falls.

Arsenic.—The output of arsenious acid in the United States in 1905 amounted to 1,545,400 lb., valued at $50,225, as against 996,456 lb., worth $29,504, in the previous year. The Everett smelter, on the Pacific coast, and the Washoe smelter at Anaconda, are the principal contributors.

Bromine.—The production of bromine in 1905 amounted to 899,434 lb., valued at $139,432, as compared with 879,312 lb., worth $215,431, in the preceding year. This was the largest annual output ever recorded. Michigan, the chief contributor, produced less bromine in 1905 than in the year before, but Ohio, Pennsylvania and West Virginia all recorded increases. About half the recorded output was recovered as bromides.

Carborundum.—The production of carborundum in 1905 fell off to 5,596,280 lb., valued at $599,628, as against 7,060,380 lb., worth $706,038, in the preceding year.

Cement.—The total output of cement of all kinds in the United States in 1905 amounted to 40,894,308 bbl., with a value of $36,012,189, as compared with 31,675,257 bbl., worth $26,031,920, in the preceding year. The output in 1905 was constituted in the following proportions: Portland, 36,038,012 bbl., valued at $0.92 per bbl.; natural hydraulic cement, 4,473,-049 bbl. at $0.54 per bbl.; and slag cement, 382,447 bbl., worth $0.71 per bbl. The total production was the largest on record. The output of Portland cement was greater in 1905 by nearly 10,000,000 bbl., while that of natural cement was 400,000 bbl. less than in 1904.

Copper Sulphate.—The production of copper sulphate in 1905 was 52,-278,996 lb., valued at $2,352,555, as compared with 63,234,557 lb., worth $3,161,728, in the preceding year. Nearly the entire output is recovered as a by-product in the various metallurgical and refining plants throughout the country; in 1905 there was no copper sulphate produced direct from ore.

Crushed Steel.—The output of this abrasive in 1905 was 812,000 lb., valued at $56,840, as against 790,000 lb., worth $55,300, in the preceding year.

Artificial Graphite.—The output of artificial graphite in 1905 was 4,439,-700 lb., valued at $303,162, as against 3,248,000 lb., worth $217,790, in the preceding year. Artificial graphite is coming to afford a very important source of the demand for graphitic carbon, although it will never replace the natural product for certain uses.

Lead Pigments.—The output of the various lead pigments in 1905 was as follows, the corresponding quantities and values for 1904 being given in parentheses for comparison: White lead, 122,398 short tons, valued at $12,068,443 (126,336 tons—$13,896,913); red lead, 16,269 short tons, valued at $1,919,767 (13,938—$1,672,569); litharge, 12,643 short tons, valued at $1,422,616 (12,487 tons—$1,248,691); orange mineral, 1000 short tons, valued at $120,000 (1125 tons—$168,681). In addition to this there was an output of 6977 short tons of sublimed white lead, consisting of sulphate and oxide of lead, valued at $697,700, as compared with 6478 tons, worth $647,800, in the previous year.

Zinc-lead Pigment.—The output of this combined pigment in 1905 was 7200 short tons, valued at $540,000, as compared with 6781 tons worth $474,670, in the preceding year.

Zinc Oxide.—The amount of zinc oxide produced in the United States in 1905 was 65,403 short tons, valued at $5,232,240, as compared with 59,613 tons, worth $4,523,414, in 1904. This remarkable increase reflects the growing favor of zinc oxide as a pigment. In addition to this there was an output of 7200 tons of zinc-lead pigment, valued at $540,000,as compared with 6781 tons, worth $474,670, in the previous year.

Metals and Alloys.

Aluminum.—We have estimated the production of aluminum in the United States in 1905 at 10,000,000 lb., valued at $3,200,000. No direct statistics are available, but this estimate is based on the knowledge of the capacity of the plants in this country. The same producers operate a plant in Canada, at which the output during the same year we have estimated at 3,100,000 lb.

Antimony.—The output of antimony in the United States in 1905 was 5,912,000 lb., valued at $614,848, as against 5,854,000 lb., worth $372,958, in the preceding year. None of this was produced from domestic ores, and much the larger part of it was never recovered as metallic antimony, but was contained in hard lead.

Copper.—The output of copper in the United States in 1905 amounted to 871,634,245 lb., as against 817,715,005 lb., in the preceding year. On the basis of the average value of Lake copper at New York, the outputs of these years were valued at $136,837,860 and $106,221,179, respectively. California was the only State to show a considerable decrease in 1905, although Wyoming and the Southern States showed a slight falling off. All other States made noteworthy increases, Arizona, in particular, showing a strong advance, having surpassed Michigan for the first time. Montana is still the largest producer, and with the ceasing of litigation in that region, a still further progress may be looked for. The price of copper toward the end of 1905 reached its highest point within five years.

Ferromanganese and Spiegeleisen.—The production of these two alloys, individual returns for which are not yet available, amounted to 289,983 long tons, valued, on the basis of the average annual quotation at Pittsburgh, at $17,639,636, as compared with 219,446 tons, worth $9,304,510, in 1904. Although spiegeleisen is worth less than ferromanganese, the above values are computed on the basis of ferromanganese, although the amount of this product is much less than that of the less valuable alloy.

Gold.—The production of gold in the United States in 1905, according to a preliminary estimate, subject to revision, by the Director of the Mint, amounted to 4,260,504 fine oz., valued at $87,948,237, as compared with 3,904,986 fine oz., worth $80,723,200, in 1904. The increase was fairly evenly distributed among the producing States, South Dakota being the only one of the heavy contributors to show a lessened output. The most noteworthy increases were recorded by Alaska, Colorado, Nevada and Utah. The total for the year was the largest annual production ever reported.

Pig Iron.—The output of pig iron in the United States in 1905, exclusive of the manganese alloys, was 22,702,397 long tons, valued, on the basis of the average annual quotation at Pittsburgh, at $377,540,862, as compared with an output of 16,277,587 long tons, valued at $225,281,804,

in 1904. More than half of this output was bessemer pig; of the remainder, foundry and forge pig and basic pig constituted about equal parts. This is the largest annual output yet reported.

Lead.—The production of lead in the United States in 1905 amounted to 322,474 short tons, as compared with 302,204 tons in 1904. On the basis of the annual average quotation for lead at New York, these outputs had the values respectively, $30,357,702, as against $26,043,941. This output came entirely from ores produced in the United States, with the exception of a few thousand tons of hard lead, of foreign, but unascertainable, origin. In addition to this, there was in 1905 a production of 83,504 tons of lead obtained from ore and bullion imported into the United States. The domestic production in 1905 comprised 205,665 tons of desilverized, 105,623 tons of soft, i.e., Mississippi Valley lead, and 11,186 tons of hard or antimonial lead, 2730 tons of which came from foreign sources. Idaho is the largest single producer, but its output is closely seconded by that of the Mississippi Valley region.

Nickel.—For a number of years, Mine La Motte has afforded a small output of nickel and cobalt. Statistics for 1905, however, are not procurable, but the output was probably insignificant. Other mines in Missouri are now preparing to yield an output of nickel. A promising discovery of ore, carrying cobalt, has been reported from eastern Oregon.

Quicksilver.—The output of quicksilver in the United States in 1905 was 30,650 flasks of 75 lb. each, as against 34,553 flasks, of the same weight, in 1904. On the basis of the annual average quotation for domestic quicksilver at San Francisco, these outputs had the values respectively, $1,189,220, as against $1,489,716. Much the largest part of this output came from the old mines of California, the working of which is gradually declining. The output of Texas was somewhat less in 1905 than in the previous year, but this output does not yet constitute a very large proportion of the total. There was a small output from some of the mines of Mercur, Utah.

Silver.—The output of silver in 1905 was 58,918,839 fine oz., as against 57,786,100 fine oz. in 1904. On the basis of the average annual quotation of fine silver in New York for those two years, these outputs had the values respectively of $35,850,955 and $33,515,938. The most noteworthy increases occurred in Arizona, Idaho and Nevada; in the other States the production was stationary or showed a decided falling off.

Zinc.—The production of spelter in the United States in 1905 amounted to 201,748 short tons, as against 181,803 tons in 1904. On the basis of the New York quotations, these outputs had the values respectively, $23,733,635, as against $18,543,906. The smelters of Kansas, which, however, draw a large part of their ore supply from Missouri, produced over half of the output in 1905. Illinois is the second largest producer.

ALUMINUM.

By Edward K. Judd.

The aluminum production of the United States can be only roughly estimated, inasmuch as the Pittsburgh Reduction Company, the sole producer of the metal, will not publish its statistics. The company has two aluminum reduction plants in this country, one at Niagara Falls, and the other at Massena, N. Y. It also has a reduction plant at Shawinigan Falls, Quebec. Most of the alumina required by these three reduction plants is refined at another of the company's works, at East St. Louis, Ill., the crude mineral coming from mines, also owned by the company, in Saline County, Ark. The works at New Kensington, near Pittsburgh, Pa., are equipped for rolling, drawing and stamping aluminum, but make no metal.

PRODUCTION, IMPORTS AND CONSUMPTION OF ALUMINUM IN THE UNITED STATES

Year.	Production.			Imports.			Exports.	Consumption.
				Crude.		Mnfrs.		
	Pounds.	Value.	Per lb.	Pounds.	Value.	Value.	Value.	Value.
1896....	1,300,000	$520,000	$0.40	698	$591	$2,888	(a)	$523,479
1897....	4,000,000	1,400,000	0.35	1,822	1,082	3,647	(a)	1,404,729
1898....	5,200,000	1,690,000	0.33	60	30	13,840	$238,997	1,474,268
1899....	6,500,000	2,112,500	0.33	53,622	(b) 9,425	7,828	291,515	1,838,238
1900....	7,150,000	2,288,000	0.32	256,559	44,455	5,989	281,821	2,056,623
1901....	7,150,000	2,238,000	0.31	564,803	104,168	5,580	183,579	2,164,169
1902....	7,300,000	2,284,590	0.31	745,217	215,032	3,819	116,052	2,387,389
1903....	7,500,000	2,325,000	0.31	498,655	139,298	4,273	157,187	2,311,384
1904....	7,700,000	2,233,000	0.29	515,416	128,350	478	166,876	2,494,952
1905....	10,000,000	3,200,000	0.32	530,429	106,108	33	290,777	3,015,364

(a) Not reported. (b) Includes manufactured. Most of the import is of crude aluminum.

We estimate the combined output of the two reduction plants in the United States during 1905 at 10,000,000 lb. and of the Canadian plant at 3,100,000 lb. These estimates are based on capacities of the plants.

Market.—The demand for aluminum is now heavily taxing the capacity of the producing plants. These are being enlarged and a new rolling mill is to be built at Niagara Falls. The shortage is partly explained by heavy buying of scrap metal for export, so that domestic consumers are obliged to use new metal. The producer asserts that it has refused many foreign orders for aluminum so as to be able to fill domestic demands more promptly. The fundamental cause of the increased demand for the metal has been probably the high price for copper, with which aluminum enters into competition in certain lines.

Prices.—The price for aluminum was stationary, at 35c. per lb. for

first-grade ingots, throughout the year until December, when an advance of 1c. per lb. occurred. The closing quotations were:

No. 1 aluminum (guaranteed over 99 per cent. pure), 35 @ 38c. per lb., according to size of order.

No. 2 aluminum (guaranteed over 90 per cent. pure, and to contain no impurities detrimental to iron and steel manufacture), 33 @ 35c. per lb. Wire of diameters 0000 to 10, 36 @ 40c., with smaller sizes ranging up to 50 @ 54c. per lb. Rolled sheets, from 44c. per lb. upward according to width and thickness.

The Aluminum Industry in Europe.

The Héroult patent, upon which the Froges method was based, has become public property. The Froges patents, covering about the same ground as the Hall patents in the United States, were the first to overthrow the Sainte-Claire Deville process, which was founded on Wöhler's classical discoveries.

Four companies now supply the European production of aluminum, their annual outputs being estimated as follows, in metric tons:

Aluminum Industrie Aktiengesellschaft	3,675 tons.
British Aluminum Company	2,250 "
Société électrométallurgique francaise	2,325 "
Société des Produits chimiques d'Alais	2,100 "

The German company has plants at Neuhausen in Switzerland, at Land-Gastein in Austria and at Rheinfelden in Baden, and employs 88 officials and 661 workmen. It paid in 1905 an 18 per cent. dividend, and recently raised its share capital from 16 to 26 million francs. The new shares were taken by a bankers' syndicate at 2.5 times par value. The Neuhausen branch has secured land at Chippis, in the Swiss canton of Valais, where it has the use of 50,000 h.p. of water power, and will erect new works. It has also purchased the chemical works of H. Bergins & Company in Deutsch Lissa, Silesia, where about 200 workmen are employed, mainly in making salts of aluminum.

The British company has works at Foyer, in Scotland, and at Milton, in Staffordshire, and uses 8,000 h.p. Another of its plants is at Sarpsfos in Norway. The company is contemplating additions to its works.

The French electrometallurgic company has plants at Froges, at La Praz and at Saint-Michel-de-Maurienne, with a combined energy of 25,000 h.p. The company has increased its capitalization repeatedly, until it now stands at 8,000,000 francs. The Société des Produits chimiques has plants at Calypso and at Saint-Félix in Savoy. It has just raised its capital to 6,000,000 francs to provide for the enlargement of its Calypso works by 6,000 h.p., to complete the purchase of the Saint-Félix works and to build a third plant, of 15,000 h.p., at Saint-Jean-de-Maurienne.

A fifth producer, details of which are not available, has recently erected

a plant in the Pescara valley of Italy, to utilize the bauxite of Lecce di Marsi.

Progress in the Metallurgy of Aluminum.

American practice[1].—The metallurgy of aluminum reverses the procedure by which all other commercial metals are obtained, for the reason that it is more difficultly reducible from its oxide than are the other metals, silicon, iron and titanium, associated with it in bauxite, as yet its only available ore. It is obvious, then, that any reduction treatment of crude bauxite that will yield aluminum will also reduce the contaminating metals, and these will destroy all the valuable qualities of the aluminum. Up to this time, no process has proved capable of removing these objectionable impurities from metallic aluminum. The procedure, therefore, is to purify the crude bauxite and then to reduce the refined alumina to metal.

The experiments of Charles M. Hall, and his later efforts to apply his discoveries to the production of aluminum on a commercial scale, form a topic of absorbing interest, and may be briefly outlined. His earliest attempts, at Oberlin College, Ohio, convinced him that electrolysis from aqueous solution was out of the question, because the nascent aluminum at the cathode was instantly oxidized. This suggested the use of organic salts devoid of oxygen as solvents, but without success. More recent experimenters, however, have proved the suitability of methyl bromide for this purpose, and have deposited aluminum from solution of aluminum bromide in this solvent. Turning then to the easily fusible halogen salts of alumina, as solvents of alumina, from which electrolysis would reduce the metal, Mr. Hall found that fused cryolite, the double fluoride of aluminum and sodium, would dissolve the oxide. At a temperature of 850 deg. C. fused cryolite will dissolve 20 to 25 per cent. of its weight of alumina, the solution taking place as quietly and effectually as that of sugar in water. The electrolysis is then a comparatively easy matter.

In the meantime, Héroult, of France, had developed the same idea, but the United States Patent Office overruled his claim to a patent in this country and Hall's patents were secured.

Hall then spent three years at the Cowles plant at Lockport, N. Y., in an unsuccessful attempt to commercialize his processes. With the Pittsburgh Reduction Company he was more successful, and in 1889 began to turn out 50 ℔. of aluminum per day; from this modest beginning, the output has advanced by giant strides. This company now holds the absolute monopoly of the aluminum-making industry in this country.

Electric power is the chief item of expense in aluminum reduction. In order to attract a desirable customer, the Niagara Power Company, which had just completed its first installation at Niagara Falls, agreed to supply

[1]Joseph W. Richards, at the Bethlehem meeting of the American Institute of Mining Engineers, February, 1906.

the Pittsburgh Reduction Company with a low-tension, direct current, for approximately $18 per horse-power per year, and to stand the expense of transformers to provide such power from its regular high-tension, alternating current. The requirements of the Pittsburgh Reduction Company's upper works have grown from 1800 to 4000 h.p., and of the lower works from 3000 to 10,000 h.p., and arrangements are now completed for 17,000 h.p. more. At Shawinigan Falls, Quebec, the same company is now utilizing 5000 h.p., and at Massena, N. Y., 2000 h.p., with the option of enlarging its consumption at the latter place to 20,000 h.p. At New Kensington, near Pittsburgh, Penn., the Pittsburgh Reduction Company operates a rolling and stamping mill for making aluminum sheets, rods, wire, tubes, and utensils, and employs 450 men. Its annual output of aluminum from all its reduction works is about equal to that of the rest of the world combined.

A process for purifying bauxite is also one of Hall's inventions. It consists in treating the impure bauxite, mixed with a proper amount of carbon, in an electric furnace, whereby the more easily reducible iron, silicon and titanium are extracted, leaving the alumina. The furnace is circular, with its walls formed of mixed carbon and bauxite enclosed between two concentric annular sheets. The bottom is a block of carbon. Carbon rods project downward into the top and can travel up or down. These carbons and the carbon bottom form the electrodes of the furnace.

The crude bauxite is first calcined in an ordinary furnace to expel water and some oxygen. It is then mixed with a little more than the amount of carbon necessary to reduce the impurities, and fed slowly into the top of the furnace, in among the carbon rods which at the beginning of operations are lowered nearly to the carbon bottom. An alternating, 1000-kw. current, at 2200 volts, is transformed to between 30 and 75 volts, capable of giving a secondary current of 33,500 amperes. The impurities (iron first, then silicon and finally titanium) are reduced and fuse into an alloy which sinks to the bottom; at the usual working temperature, 3000 to 3500 deg. C., a little aluminum also is reduced, while the purified bauxite contains a small amount of aluminum sub-oxide. As the melt accumulates the carbon rods are withdrawn so as to leave but a short gap above its surface, and the bauxite mixture is added continuously until the furnace is full of the fused mass. The excessive heat of the fusion makes it impossible to tap the furnace; then, too, if the fused alumina were cooled quickly it would form densely hard artificial corundum, unsuitable for aluminum reduction. The entire furnace is therefore carried outdoors and its 15 tons of contents allowed to cool slowly. When cold, the alumina is found crystallized into an easily granulated saccharoidal mass, of a lilac color.

The earliest analysis of this material showed no iron or silicon, a trace

of titanium and over 100 per cent. of alumina. The alumina having been weighed as Al_2O_3, this result suggested the presence of a lower oxide, possibly Al_3O_4, which was confirmed by microscopic study. The substance was then found to consist of a ground mass of Al_3O_4, constituting 8 to 10 per cent. of the whole, enclosing minute crystals, of Al_2O_3, which the goniometer showed to have the characteristic angles of corundum. No metallic aluminum is present. This granular alumina is even more suitable for reduction than real Al_2O_3, since its weight enables it to be handled and dissolved more easily than the light amorphous powder.

The aluminum reduction furnace is a heavy cast-iron trough, lined with carbon and having a depression for holding the bath. It is blocked up from the floor so as to permit cooling ventilation. A group of 48 carbon rods, each 3 in. in diameter and 15 in. long, projects down into the depression and forms the anode, the lining of the cast-iron trough itself being the cathode.

In operation the trough is nearly filled with a fused bath of cryolite and a little aluminum fluoride, heated to a cherry red, at 850 to 900 deg. C. On the bath floats a thin layer of broken pieces of carbon. On this the refined alumina is scattered to dry and warm, and is made to fall through into the bath from time to time by stirring the floating layer. The carbon anodes project nearly to the bottom of the trough. Each carbon takes 200 amperes, equivalent to 30 amperes per square inch of trough surface, or 10,000 amperes for one furnace. Electrolysis absorbs 2.2 volts, and resistance 3 to 3.5 volts; the total electromotive force required is about 5.5 volts.

The melted aluminum collects in the bottom of the furnace and is tapped off at the side through a conical hole. This is closed by a plug of carbon and pitch, inserted on the end of a pointed bar. The yield from such a furnace amounts to about 1.75 lb. per horse-power day. It is essential to keep the bath heated to its working temperature, since, by cooling, the relative densities of the aluminum and the bath are reversed, as is shown by the following comparison of specific gravities:

	Cold.	Hot.
Bath	2.9	2.3
Aluminum	2.6	2.5

If cooling occurs, the aluminum tends to rise above the bath and short-circuits the current. The same thing will occur if alumina is fed in too rapidly.

The Betts Aluminum Process.—Alumina occurs in nature, combined with the oxides of iron, and silicon in clay, and mixed with those of iron, silicon and titanium in bauxite; but since those oxides are more easily reduced than alumina, it has not been possible to obtain aluminum by

reducing the ore to an alloy and slagging off the impurities, as with iron. For this reason the practice has been to free alumina from impurities before reducing it to metal. Anson G. Betts, of Troy, N.Y.,[1] proposes a process radically different in principle. He reduces the crude ore direct to a metallic state, either with or without diluting the aluminum by the addition of another metal, as copper, zinc or tin. The product of aluminum, mixed with other metals, is placed in a fused state at the bottom of an electrolyzing cell, containing an aluminum-depositing electrolyte which is specifically lighter than the aluminum-containing product, and floats on it, being also specifically heavier than pure fused aluminum. A layer of pure fused aluminum is electrically connected as cathode, while the aluminum-containing product is electrically connected as an anode. The property of aluminum, as the most easily oxidized of metals, insures that it alone will dissolve from the anode, so that, with the passing of the current, pure aluminum is deposited at the cathode, while the deficiency of aluminum caused thereby in the electrolyte is made up by solution from the anode.

It is claimed that by this method aluminum can be extracted from such materials as aluminum silicide and carbide and similar non-oxygenated compounds. It may be applied to the reduction of bauxite (similarly to the Hall process for aluminum) from a bath of fluorides; also to the reduction of bauxite, clay or other aluminous material, by alloying the product with iron, copper, tin, etc., and extracting the contained aluminum electrolytically, leaving the other elements alloyed with the heavy metal in question.

This method has not yet been tried outside of the laboratory.

Dehydration of Alumina.—The Société des Produits chimiques d'Alais[2] states that complete dehydration, and at a lower temperature than usual, is obtained by mixing 1 per cent. aluminum fluoride with the hydrate of alumina, and that this treatment has the effect also of preventing rehydration from atmospheric moisture.

Reduction of Aluminum from Clay.—Clay is by far the most abundant aluminum-containing substance in nature, and many attempts to recover the metal have been made. The General Electric Company, of Schenectady, N. Y., proposes[3] to mix clay with coke and heat the mixture in the electric furnace to form carbides of aluminum and silicon. The aluminum carbide may then be leached with a caustic soda solution and separated from the carborundum.

Extraction of Alumina from Feldspar.—The feldspars are alkaline silicates of aluminum, and from these G. Levi, of Rome, Italy, proposes[4] to

[1] United States Patent No. 795,886.
[2] British Patent No. 7,032 of 1904, Aug., 1905.
[3] British Patent No. 3,998 of 1904, Jan., 1905.
[4] British Patent No. 13,875 of 1904, May, 1905.

extract the alumina by digesting them under high pressure with alkaline solutions, so as to form alumina and alkaline silicates.

Soldering.—The rapidity with which a fresh surface of aluminum becomes coated with a tenacious and impervious coating of oxide is the obstacle to aluminum soldering. As yet no flux has been found which will dissolve this oxide coating and keep the metallic surface fresh sufficiently long to permit soldering in the ordinary way. A satisfactory method is said to consist in using a heavy, untinned soldering copper, heated to dull redness. If then the aluminum surface is scratched vigorously with the copper, the solder meanwhile being melted against it, the oxide coating will be removed mechanically and give the solder an opportunity to attach itself. Aluminum must be heated to 660 deg., or 200 deg. higher than required by brass, before solder will adhere to it. Two pieces of aluminum are joined by first coating each of the abutting faces with solder, in the above way, and then bringing them together under heat and pressure. A solder recommended for this kind of work has the composition, by weight: Aluminum, 1; phosphor-tin, 1; zinc, 11; tin, 29. The aluminum is melted first, in a crucible, and the zinc, tin, and phosphor-tin are added slowly in the order named. The fusion is stirred with a brass rod and poured into sand molds. The solder has not the same color as aluminum and cannot, therefore, be used in ornamental work.

Other solders for aluminum have been patented during the year, one of which has the composition: Tin, 64; zinc, 30; lead and aluminum, each 1 part.[1]

Another contains silver, aluminum, zinc and tin. A portion of the aluminum, silver and tin must be phosphorized and a part of the zinc sulphated, but the amount of each element so altered is not stated.[2]

An autogenic method consists in cleaning the parts to be joined, fitting them together, and heating the joint to a temperature just below the fusing point of aluminum. Then a rod of metal, having a fusion temperature somewhat higher than that of aluminum, is applied, and the heat is raised to the melting point of aluminum.[3]

Chr. Sorensen, of Siagelse, Denmark, suggests cleaning the surface of the metal with hydrochloric acid and covering it with powdered potassium chloride, which is then carefully melted. The surface may then be soldered with metallic tin.

Aluminum as a Pattern Metal.—For many kinds of patterns, aluminum has advantages over other metals used for this purpose. Its specific gravity is only one-third that of brass, or of the tin-zinc-antimony alloys commonly used, whereby it is easily and safely handled in large patterns

[1] F. Jackman, Bradford, England. British Patent No. 17,031 of 1904.
[2] Fortun and Sempruns, Madrid, Spain. United States Patent No. 778,025.
[3] Massimo Tomellini, Genoa, Italy. United States Patent No. 783,332.

and its cost, per unit volume, is small. The objection that molding sand sticks to aluminum more than to wood or to other metals seems to be without foundation. Aluminum is more easily finished by file, sandpaper or scraper than brass or the hard metals, and casts more smoothly than the soft alloys. Its rigidity prevents its being bent or sprung, but it is sufficiently ductile to be trued up if deformed.

It is worthy of note that the form of grate now used in the Wetherill zinc-oxide furnace is cast in sand molded by an aluminum pattern. This is a very delicate casting, for which the preparation of a wood pattern involved a multiplicity of small pieces, rendering it both difficult to make and expensive. The aluminum pattern has proved a great improvement.

Plating on Aluminum.—Aluminum is prepared to receive a plating of any metal by first giving it a thin coating of tin, in the process patented by J. Creswick and H. Shaw of Sheffield.[1] Metallic tin is first deposited upon the article by immersing it in a solution of stannous chloride and ammonium alum, prepared by dissolving 2 oz. of the former and 5 lb. of the latter in water. The plating is afterwards carried out in the usual way.

La Metallurgie states[2] that the difficulty in the way of plating aluminum lies in obtaining an absolutely clean surface for deposition. Clean scouring may be accomplished by dipping into a hot 10 per cent. solution of potash, rinsing and dipping into a cold mixture of 5000 grams nitric acid, 250 grams sea salt and 25 grams lampblack. After this treatment, the aluminum may be plated electrolytically in baths composed as follows: *Gold*—cyanide of gold, 10 grams; sodium bisulphite, 750 grams; sodium phosphate, 900 grams; water, 10 liters. *Light yellow copper*—sodium bisulphite, 350 grams, and potash, 500 grams, dissolved in 10 liters of water, to which is added acetate of copper, 175 grams; zinc chloride, 175 grams; ammonia, 200 grams in 2 liters of water. This bath, if heated during electrolysis, gives a beautiful golden-yellow coating. Another consists of zinc sulphate, 150 grams; copper sulphate, 150 grams; sodium bisulphate, 200 grams; sodium carbonate, 200 grams dissolved in water. Each of these baths requires an electromotive force of 4.5 volts and an intensity of 60 amperes, factors which should be carefully maintained. *Red copper*—copper sulphate, 5 kg., water 100 liters, electrolyzed at 2 volts and 60 amperes. Aluminum plating on iron or copper may be done with an aluminum anode in a bath containing 175 grams sodium bisulphite, 75 grams zinc chloride, and 200 grams ammonia in 25 liters of water.

Alloys.—Aluminum bronzes are useful in machine (particularly automobile) design, where strength and lightness must be combined, and in decorative work where a yellow non-tarnishing metal is desired. M. Guillet has made exhaustive tests on the physical properties of aluminum

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[1] British Patent No. 21,609, Oct. 8, 1903.
[2] Dec. 7, 1904, p. 2311.

bronzes.[1] These are of two classes: Those high in copper, containing up to 10 per cent. aluminum, and those rich in aluminum, containing from 3 to 6 per cent. copper. The first are known as aluminum bronzes, while the second are sold under the name of aluminum, and are utilized in the construction of parts for automobiles. In preparing these alloys, protection from oxidation is necessary and castings are best poured by means of a plug head. The high contraction must also be allowed for. Up to 10 per cent. aluminum the alloys may be rolled, but when the aluminum exceeds 7 per cent. drawing is difficult. In color the alloys are golden up to 14 per cent. aluminum, but as this is exceeded they become white, gray and silver-white. Up to 10 per cent. aluminum the alloys are malleable, but at 11 per cent. they become fragile. The difficulty of preparation also increases as the alloys increase in aluminum. Malleability only returns when the

TABLE I.

Alloy.	Elastic Limit, lb. per sq. in.	Max. Stress, lb. per sq. in.	Elongation, per cent.	Alloy.	Elastic Limit, lb. per sq. in.	Max. Stress, lb. per sq. in.	Elongation, per cent.
Cast.				Aluminum, commercial, annealed...	6,868	15,642	23
Aluminum commercial............	4,266	13,367	7	Aluminum, commercial, cold worked	7,252	15,926	11
Aluminum + 1% copper	4,977	14,647	5	Alloy at 2.5% copper	8,959	28,156	11.5
Aluminum + 2% copper	5,688	15,500	4.5	Alloy at 3.6% copper	9,385	30,431	20
Aluminum + 3% copper	6,968	17,064	5	Alloy at 91.95% copper	61,000	65,696	43
Aluminum + 4% copper	7,394	18,344	6	Alloy at 92.23% copper	49,340	61,430	54
Aluminum + 5% copper	7,110	18,770	3	Aluminum, pure, annealed		17,064	31
Aluminum + 6% copper	7,394	19,339	2				
Cast:				Aluminum bronze, cold worked.....		22,752	
Copper + 3% aluminum..........	11,234	29,720	36	Alloy at 3% copper annealed......		27,018	9.5
Copper + 5% aluminum.........	11,945	33,986	53	Alloy at 3% copper, cold worked...		31,284	4.5
Copper + 7% aluminum.........	12,798	36,119	66	Alloy at 6% copper, annealed.....		28,440	11
Copper +10% aluminum	24,032	43,371	6	Alloy at 6% copper, cold worked..		35,550	4

TABLE II.

Composition.					Tensile Properties.		
Cu.	Al.	Fe.	Si.	S.	Elastic Limit, lb. per sq. inch.	Max. Stress, lb. per sq. inch.	Elongation, per cent.
On Cast Bars.							
99.74	..	..	..	..	12,798	30,715	26.7
93.35	4.62	0.89	0.98	0.04	13,793	53,182	46.6
91.17	5.92	0.78	2.12	0.04	22,325	70,530	28.1
90.43	7.50	0.54	1.54	0.05	20,190	69,100	31.3
89.67	7.08	0.72	2.72	0.12	31,568	75,500	7.4
86.07	10.05	0.98	2.48	0.14	.....	89,440	0.15
On Rolled Bars.							
90.99	7.96	....	1.36	....	30,715	70,958	18.4
90.38	8.29	....	1.41	0.04	35,690	76,646	15.3
89.77	7.43	0.54	2.53	....	35,834	82,334	16.5
91.66	6.69	....	1.83	0.04	33,133	74,655	33.5
90.38	7.62	0.73	1.44	....	30,000	73,944	25.3
88.83	7.19	2.27	1.32	0.05	28,156	74,228	35.4
93.59	4.99	0.91	0.91	....	24,600	62,140	41.6
89.99	7.98	0.89	1.23	....	31,000	72,379	12.5
86.71	9.80	0.78	2.38	....	44,500	98,970	0.4
89.98	7.37	1.02	1.95	....	29,290	78,780	18.3
88.16	11.01	0.34	0.80	....	44,220	72,660	0.2
90.50	8.81	0.56	0.01	....	31,710	72,100	20.7

[1] *Revue de Metallurgie*, 11, 8.

aluminum exceeds 66 per cent. Tensile results obtained from the alloys in various conditions are given in the foregoing table I.

The result of quenching appears to be to increase breaking load and elongation, an effect more pronounced in cast than in rolled material. Complex aluminum bronzes are alloys to which an addition such as silicon has been made with a view to improving the quality. A series of these complex bronzes is given in table II, on the preceding page.

New Uses.—Aluminum foil is used largely as a substitute for tin foil. It is said that 156 sq. ft. of thin sheets may be spun from 1 ℔. of the metal. Those sheets which adhere together are worked into powder.

Spools and bobbins, particularly for use in mills, are now made of aluminum. They weigh about one-half as much as wood, are less influenced by heat and moisture, and are said to be more durable.

Aluminum is used in the manufacture of certain explosives. One, for example, consists of ammonium nitrate, 45 parts; di- or trinitrotoluene, 1905 parts; aluminum, 22 parts.

Aluminum Casting.—By observing a few precautions, aluminum may be cast as easily as other metals. It melts at about 1200 deg., and absorbs gas when overheated, or when kept molten a long time. A large sprue and gate should be used to furnish metal as the casting shrinks, and they should be located at the heaviest part of the casting. The use of risers is important if a casting is large. The sand should be tempered as dry as can be worked, to avoid blows, and if the mold has pockets of sand, the drag should be vented to the bottom board. The sand should be rammed as lightly as possible. The metal should be poured at the lowest temperature which will allow filling the mold—generally just above the melting point—and pouring should be done as rapidly as possible.

Aluminothermics.[1]—The welding of rails by the thermit process is becoming more popular in this country, 30 or more cities having tested it, some of them, as Cleveland, O., and Holyoke, Mass., on an extensive scale; about 13,000 joints have been made since the introduction of the process. The Goldschmidt Thermit Company, which controls the process, now has a plant in this country for manufacturing the thermit mixture and other supplies, which has been at work since 1904, and the importation of foreign-made material has ceased. The process of rail welding consists in clamping the two halves of a sand-lined mold around the abutting ends of the two rails. A refractory-lined crucible is then mounted, on a tripod, with its pointed end over the gate of the mold, and is charged with the necessary amount of thermit, a finely pulverized mixture of aluminum and iron oxide; fragments of steel may be added to increase the yield of metal and to prevent an unnecessarily high temperature.

[1] From *Journal* of the Franklin Institute, Dec. 1905, p. 435. E. Stütz.

The mixture is ignited by a match, through the medium of an ounce of an aluminum-barium peroxide mixture, and, when the reaction is complete, the resulting iron is tapped from the crucible and allowed to fill the mold. The gate of the mold is so arranged as to bring the fluid iron up from under the rail and to form a collar around the joint. Only the alumina slag comes into contact with the head of the rail, bringing the whole section to a uniform temperature and preventing oxidation and softening of the rail head. In fact, the steel at the weld appears harder than at a distance from it. About 10 ℔. of steel, heated to 5400 deg. F., suffices for a rail weld.

The mold is the most important part of the apparatus. It consists of a sheet-iron box into which equal parts of sharp sand and brick clay, or sand and flour, are molded so as to fit the rail. A sand-clay mold must be dried at a high heat to remove every trace of moisture; with a sand-flour mold this is not necessary, but this lining is applicable only to the smaller jobs. The mold must fit the rail section accurately, and when in use must be luted tight with clay; a leak would not only waste the thermit steel, but the flow would carry with it the fused substance of the rails.

Thermit welding has been applied, with marked economy, to the repairing of locomotive frames, fly-wheels, and the steering and propelling mechanisms of steamships, the usual long and expensive dismantling being escaped in every instance. The amount of thermit required by any unaccustomed repair is a matter of judgment. In general, the amount of liquid steel provided should be twice that required to form the collar around the break. This assures a free circulation of heated metal through and around the fracture. The collar should be allowed to remain wherever it is at all possible, so as to give added strength.

To weld pipes, without supporting the interior, requires special procedure. The ends are clamped in abutment and a simple sheet-iron box surrounds the joint. The reaction takes place in a flat-bottomed crucible, and the slag is poured first; this prevents the succeeding metal from melting into the walls of the pipe and a butt-weld is obtained in one and one-half minutes.

In the foundry, thermit containing a little titanium may be introduced into the bottom of a ladle of cast iron. Owing to the affinity of titanium for nitrogen, air is freely evolved from the iron and a denser, finer grained casting results. The temperature is not raised, since the thermit weighs only 0.06 to 0.25 per cent. of the iron in the ladle. Some results of tests made by the Riehlé Bros. Testing Machine Company upon malleable cast-iron bars made by the Pennsylvania Malleable Company at McKees Rocks, Penn., show the influence of titanium thermit. The bars were cast from the same ladle, half of them before, and half of them after the addition

of the titanium mixture. The span over which the deflection was observed is not noted.

TESTS OF MALLEABLE CAST-IRON BARS.

Before Reaction.				After Reaction.			
No.	Section In.	Ult. Strength Lb.	Deflection In.	No.	Section In.	Ult. Strength Lb.	Deflection In.
1	1.000 x 0.999	4,100	1.00	1	1.011 x 1.010	5,920	1.30
2	0.995 x 0.999	4,500	0.98	2	0.999 x 1.000	4,260	1.27
3	1.060 x 0.998	4,540	1.28	3	0.989 x 0.995	4,850	1.55
4	1.012 x 1.006	4,610	1.40	4	0.995 x 0.996	4,620	1.47
5	1.006 x 1.005	4,500	1.40	5	0.998 x 0.996	4,410	1.37
				6	1.011 x 1.000	4,810	1.44
Aver.		4,450	1.21	Aver.		4,811	1.60

Metals and alloys free from carbon are being produced on a commercial scale by alumino-thermic methods. Manganese and chromium are the chief metals so produced. It is found that the purer these metals are, the better the results obtained from their use in the manufacture of special steels. It has been determined that instead of using exactly equivalent amounts of the oxide and of aluminum, it is advisable to have more oxide and less aluminum in the mixture. By suitably regulating the proportions it is possible to obtain metal free from aluminum.

ANTIMONY.

By Edward K. Judd.

Practically the entire production of metallic antimony in this country is still derived from imported ores. A few shallow workings in antimony ores, located at isolated points in California and Nevada, make sporadic shipments to the antimony smelters, mainly to the Chapman company of San Francisco, but European and Australian ores, entering duty free, supply the entire demand for raw material in the East. The smelters also find it cheaper to import impure metal and partially refined stibnite, paying a duty of $\frac{3}{4}$c. per pound, and to refine it in this country.

The principal smelter is Mathison & Co., whose plant in this country is at Chelsea, Staten Island, N. Y. This smelter made a small output of antimony from domestic ores in 1905, which it marketed under the "U. S." brand. The company refuses to report its production, wherefore direct statistics of the production of this metal cannot be given.

The production may be closely approximated from the net importation of ore. Estimating an average yield of 40 per cent. from this, the antimony production of the United States in 1905 was 395 short tons, against 412 tons in 1904.

Imports and exports of antimony are shown in the following table:

ANTIMONY STATISTICS OF THE UNITED STATES.
(In short tons.)

Year.	Imports.		Exports.		Production.			Consumption
	Metal or Regulus.	Ore.	Metal or Regulus.	Ore.	In hard Lead.	From Domestic Ore.	From Imported Ore. (a)	
	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons
1896...	1,792	590	....	....	1,877	....	236	3,905
1897...	573	2,751	....	....	2,217	245	1,100	4,135
1898...	1,013	1,863	13	17	2,118	250	738	4,106
1899...	1,580	1,991	*Nil.*	*Nil.*	1,586	234	796	4,196
1900	1,816	3,018	12	*Nil.*	2,476	151	1,207	5,638
1901........	1,837	866	*Nil.*	25	2,235	60	336	4,458
1902........	2,871	840	37	104	2,904	*Nil.*	294	6,032
1903........	2,563	1,337	40	*Nil.*	2,552	*Nil.*	535	5,610
1904........	2,028	1,245	16	214	2,515	*Nil.*	412	4,939
1905....	2,869	988	*Nil.*	*Nil.*	2,561	*Nil.*	395	5,825

(a) Estimated at 40% extraction from net imports of ore.

From this it will be seen that in 1905 the metal yielded by the duty-free ores was less than one-seventh of the amount of the metal on which duty was paid.

Market.—The shortage of antimony occasioned by the shutting off of supplies from Japan made itself severely felt during 1905. This condition was aggravated by the increased consumption, partly for war purposes

but largely for the industries, so that by the end of June prices which, at the beginning of the year, had been 8½c. per lb. for Cookson's, 8¼c. per lb. for Hallett's, and 8c. per lb. for other brands, had advanced 3c. per lb.

During July European producers again raised their prices, which at the end of August stood at from 14 to 15½c. per lb. Subsequently prices declined, as it was expected that, the war being over, Japan would again enter the market with its production. On this downward movement the lowest prices were reached about the middle of November, when quotations were 12c. per lb. for Cookson's and 10½c. per lb. for other brands. As exports from Japan, however, failed to materialize, prices again advanced rapidly, and stood at the beginning of December at 13½c. per lb. for Cookson's and 12½c. per lb. for other brands.

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK.
(Cents per pound.)

	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year
1901.													
Cookson's	10.00	10.00	10.00	10.31	10.25	10.25	10.25	10.25	10.12	10.09	10.00	10 00	10.12
Hallett's	9.12	9.22	8.90	8.94	8.75	8.75	8.75	8.43	8.50	8.47	8.37	8.31	8.74
Others	9.25	8.85	8.77	8.73	8.63	8.63	8.63	8.50	8.37	8.34	8.25	8.00	8.55
1902.													
Cookson's	10.00	10.00	9.87	9.87	9.87	9.87	9.75	9.75	9.69	9.44	9.25	9.20	9.71
Hallett's	8.17	8.04	8.06	8.06	8.17	8.25	8.25	8.15	7.92	7.72	7.44	7.25	7.96
Others	7.86	7.75	7.75	7.75	7.90	8.00	8.00	7.90	7.65	7.37	7.22	6.92	7.67
1903.													
Cookson's	8.25	8.25	8.25	8.25	8.00	7.50	7.44	7.15	7.00	7.00	6.56	6.75	7.53
Hallett's	7.00	7.00	6.87	6.87	6.75	6.69	6.50	6.40	6.34	6.25	6.25	6.35	6.69
Others	6.75	6.62	6.50	6.50	6.50	6.44	6.25	6.19	6.00	6.00	6.00	5 95	6.31
1904.													
Cookson's	6.938	7.594	7.875	7.875	7.531	7.200	7.188	7.188	6.913	6.984	7.592	8.388	7.439
Hallett's	6.250	6.781	6.825	6.750	6.578	6.438	6.485	6.688	6.537	6.578	7.328	8.160	6.783
Others	5.688	6.203	6.475	6.406	6.203	5.961	5.969	6.062	6.015	6.172	7.204	8.088	6.371
1905.													
Cookson's	8.375	8.375	8.375	8.219	8.406	11.025	12.625	14.500	13.700	13.000	12.500	14.000	11.100
Others	8.063	8.063	7.638	8.125	8.406	10.175	11.875	13.500	12 900	12.000	11.250	12.750	10.400

The pronounced shortage of stocks made itself felt during December, and as prices were advanced by Cookson's from time to time, the market here was quoted very much higher; 15½c. was asked for Cookson's antimony, while other brands could be procured at prices ranging between 13½c. and 14½c. per lb.

Progress in the Metallurgy of Antimony.

Much inventiveness has been applied to the extraction of antimony from its ores or impure bullion by leaching processes, followed by electrolysis for recovering the metal. German investigators have tried chloride solutions for leaching ores or bullion, and alkaline sulphide solutions for ores alone. The electrolytic refining of impure antimony bullion is worthy of attention for the high value in gold which this sometimes contains.

Betts' Investigations.—Anson G. Betts has reported[1] some of his investigations as to the electrolysis of antimony solutions. Acid ferric chloride solution is said to attack stibnite with ease, producing a solution of ferrous and antimonious chlorides and a residue of sulphur; from this solution metallic antimony and ferrous chloride can be recovered by electrolysis. The same idea has been applied to the treatment of crude

[1] Meeting of the American Electrochemical Society, Sept. 18, 1905.

antimony, and it has been found that the electrolytic operation can be easily carried out.

Carbon anodes and copper or lead cathodes may be used. The ferric chloride generated at the anode is heavier than the main solution, and continually runs down the anode surface and collects at the bottom of the cell, whence it may be drawn off, while fresh ferrous antimonious chloride is added at the top. The solution should be strongly acid to prevent the precipitation of white antimony oxychloride.

Antimony trifluoride is much superior to antimony trichloride for electrochemical purposes; oxygen is liberated at the lead anode and hydrofluoric acid is formed in the electrolyte. When iron is present a good deal of ferric salt is formed, particularly with low anode current densities.

Antimony oxide is obtained in some metallurgical operations, such as the roasting of antimony sulphide, and this may be dissolved in hydrofluoric acid and electrodeposited from the solution with lead cathodes and lead anodes as a beautiful solid deposit of high purity, if the solution is free from copper.

Copper is the most troublesome metal in the electrometallurgy of antimony. In the chloride solution it stands very close to antimony in the series, but in the fluoride solution it stands lower in the scale, and may be precipitated from solution by antimony.

When the fluoride solution was first tried for the electrolytic refining of antimony, alkaline sulphates and sulphuric acid were added to improve the conductivity, and in the hope of making the use of lead linings even more successful.

Beautiful deposits were obtained, but if the anode contains even very small amounts of impurity, the polarization rapidly increases, in some cases almost stopping the current entirely, and always causing the solution of impurities of the anode, such as copper, so that no purification is effected from copper.

In such a solution the current would be mainly carried by the sulphuric acid, and the anions liberated would be almost entirely sulphion, so that antimony sulphate would be primarily formed. Antimony sulphate would immediately decompose into basic sulphate, which would then have to be dissolved by the hydrofluoric acid present. If the anode contains lead, for example, there would be enough lead sulphate present to interfere seriously with the action of the hydrofluoric acid. It is therefore concluded that the cause of the trouble is the presence of sulphion, and experiment shows this to be a fact.

With the omission of sulphion there is no difficulty at all. The anodes dissolve as readily as copper or lead anodes, and the cathode deposit of antimony leaves nothing to be desired. No analyses have as yet been

made of the antimony, but there is no reason to think that it is not exceedingly pure.

MacArthur Process.—J. S. MacArthur proposes to treat ore containing antimony, ground to 20-mesh, with a solution of sodium hydroxide, of not more than 2 per cent. strength, unless the ore contains oxide of antimony, at a temperature of 100 deg. C. The filtered solution is precipitated by carbon dioxide obtained from a lime kiln, the reaction being assisted by cooling. The antimony comes down as a red precipitate of amorphous sulphide, which is then collected in a filter-press, and the filtrate, containing sodium carbonate, is causticized by lime from the same kiln for further use, thus forming a cyclic process.

This treatment is now being applied successfully by the Dominion Antimony Company, in Nova Scotia, to the antimony-gold ore of West Gore, Hants county. These lodes yield an ore in which gold is closely associated with both native antimony and stibnite. After sorting, the first grade ore carries 45 to 50 per cent. antimony and 2.5 to 3 oz. gold per ton; the second grade has 20 to 30 per cent. antimony and $20 to $30 per ton in gold. The ratio between the quantities of the two metals is fairly constant. The company hopes to find a method of removing the soda in the residue from the antimony treatment so as to be able to treat this sand by the cyanide process to recover its gold. At present the selected gold ore is sent to Swansea.

S. Metzel proposes to prepare antimony oxide from crude sulphide by warming the powdered ore with sulphuric acid in the presence of sodium or potassium sulphate. A crystalline double sulphate of antimony and the alkali-metal separates on cooling; on boiling this with water, it is decomposed and the antimony oxide is left, releasing the acid, which is returned to the process.

Explosive Antimony.—E. Cohen[1] discusses the nature of an explosive form of antimony which is deposited by electrolysis, under certain determined conditions of temperature and concentration, from trichloride or other antimony haloid solution. The metal thus deposited will explode from a shock, giving off a white vapor which consists of haloids previously combined with the metal in a solid solution. The heat comes from the transformation of the antimony into another allotropic modification.

ANTIMONY ALLOYS.

Bearing Metal.—The principal use of antimony is its fusion with lead, and small proportions of other metals, to form type and bearing metals. For this reason antimony is not extracted from hard lead, in which form much the larger part of its output is obtained, but the compound product is used directly.

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[1] *Zeitschrift für Elektrochemie*, Nov. 10, 1905.

The specifications for bearing metal enforced by the German railroads read as follows:

"The babbitt metal is to be composed of tin, antimony and copper in the following manner: 1 kg. of copper is melted with 2 kg. of antimony and 6 kg. of absolutely pure tin. Add the antimony as soon as the copper melts and the tin as soon as both other metals are melted. This alloy is cast in thin plates and 9 kg. of it are then melted with 9 kg. of pure tin. The mixture is cast into ingots 15 mm. thick ready for use. Larger quantities than the above quoted may not be melted together at the same time. The components of this alloy must be as pure as possible. The antimony may have at the most 1 per cent. of impurites, and of these not more than 0.10 per cent. of arsenic. The tin may not contain more than 0.02 per cent. of impurities. There must be neither lead nor zinc in the mixture."

These specifications are particularly rigid. In American practice a ton or more of babbitt metal is melted with good results. The German mixture as described above gives the following percentage, which compares with the United States standard formula:

	Tin.	Antimony.	Copper.
German Railroad Specifications..................	83.3%	11.1%	5.6%
U. S. " "	88.9	7.4	3.7

The German babbitt metal is somewhat harder than that made in this country, and for some uses is probably too hard. Many railroads in this country, however, buy a hard babbitt metal and reduce it with pure tin to suit various requirements.

Analytical Methods.

Electrolytic Determination of Antimony and Tin.—Arthur Fischer reports[1] the following conclusions as to the electrolytic behavior of these metals in solutions of their sulpho-salts: (*a*) Tin can be quantitatively deposited on a tinned platinum cathode, out of a solution of tin-ammonium-sulpho-salt, with the addition of sodium sulphide to ammonium sulphide. A platinum-iridium anode is recommended. (*b*) Antimony can be deposited quantitatively, as pure metal, from either sulphantimonite or sulphantimonate, in sodium sulphide solution, with the addition of potassium cyanide. The interference of polysulphides is negligible, since they are readily reduced by the cyanide. Here, too, a platinum-iridium anode is preferred to one of platinum. (*c*) The electrochemical equivalent of antimony in sulphantimonite is $\frac{Sb}{3}$; and in sulphantimonate, $\frac{Sb}{5}$. Pentavalent antimony tends to reverse the reaction, while trivalent antimony assists it. (*d*) The separation of antimony from tin in sodium sulphide solution is as successful with trivalent as with pentavalent antimony, with the addition of potassium cyanide. The

[1] *Zeitschrift für anorganische Chemie*, 1904, p. 363, and *Chemiker Zeitung*, 1905.

solution has to be saturated, at 30 deg. C., with sodium sulphide, and must contain 2 to 4 grams of free sodium hydroxide per liter. The temperature must not exceed 30 deg., nor the electromotive force 1.1 volt. (*e*) Without potassium cyanide in the solution, only trivalent antimony can be accurately separated electrolytically from tin. In this case, the solution must be saturated, at 30 deg., with sodium sulphide, and the electromotive force must not exceed 0.9 volt. (*f*) Sodium sulphohydrate solution is not suitable for the separation, since, in this solution, the potentials of tin and antimony are but slightly different. (*g*) The addition of sodium hydroxide, recommended by Classen for the separation, is particularly desirable for minimizing the hydrolysis of unhydrated sodium sulphide. (*h*) The trisulphide, or gravimetric, method for antimony gives lower results than the electrolytic, since the antimony sulphide determined by the former does not correspond strictly to its formula.

WORLD'S PRODUCTION OF ANTIMONY ORE. (*a*)
(In metric tons.)

	1895.	1896.	1897.	1898.	1899.	1900.	1901.	1902.	1903.	1904.
Austria	695	905	864	679	410	201	126	18	41	103
Bolivia	......	......	......	......	1,213	1,174	190	126	45	......
France and Algeria	5,703	6,333	5,466	4,571	7,592	7,963	9,867	9,715	12,380	9,065
Hungary	1,240	1,361	1,800	2,201	1,965	2,373	323	748	205	108
Italy	2,241	5,086	2,150	1,931	3,791	7,609	8,818	6,116	6,927	5,712
Japan (*b*)	1,061	827	348	1,006	712	81	119	88	153	......
Mexico (*c*)	600	3,231	5,873	5,932	10,382	2,313	5,103	1,279	1,856	18
N. S. Wales (*d*)	486	135	172	84	332	252	90	57	13	111
New Zealand	55	21	10	......	......	5	30	......	......	......
Portugal	753	595	417	245	59	38	126	......	......	31
Queensland	......	......	......	......	41	......	......	......	......	......
Spain	44	54	354	130	50	30	10	67	42	245
Turkey	1,322	100	400	(*e*)	1,173	267	224	(*e*)	(*e*)	(*e*)
United States	982	136	454	(*e*)	544	300	100	......	......	......

(*a*) From official reports of the respective countries. (*b*) Mostly crude antimony. (*c*) Export figures, except for 1903, which represents production. (*d*) Metal and ore. (*e*) Not reported.

Owing to the trouble of preparing the sodium sulphide solutions necessary for electrolyzing antimony solutions by the method of Classen, H. D. Law and Dr. F. Mollwo Perkin recommend the use of a solution containing ammonium tartrate, which they have found to give satisfactory results.

ARSENIC.

The United States in 1905 made 1,545,400 lb. of white arsenic, valued at $50,225; an increase of 548,944 lb., or 55 per cent., over the output of 1904. Not all of this was marketed.

The first production of white arsenic in this country occurred in 1901. Before that year, consumption was supplied by importation from Europe; and since domestic production began, imports have supplied still nine-tenths of the white arsenic used in this country, as is shown by the table given herewith.

ARSENIC STATISTICS OF THE UNITED STATES

Year.	Production.			Imports.			Consumption.	
	Pounds.	Value.	Per lb.	Pounds.	Value.	Per lb.	Pounds.	Value.
1896	....	....	....	5,813,387	$215,281	0.03¾	5,813,387	$215,281
1897	....	....	....	7,242,004	352,284	0.05	7,242,004	352,284
1898	....	....	....	8,686,681	370,347	0.04¼	8,686,681	370,347
1899	....	....	....	9,040,871	386,791	0.04¼	9,040,871	386,791
1900	....	....	....	5,765,559	265,500	0.04¾	5,765,559	265,500
1901	600,000	$18,000	$0.03	6,989,668	316,525	0.04½	7,589,668	334,525
1902	2,706,000	81,180	0.03	6,110,898	280,055	0.04½	8,816,898	361,235
1903	1,222,000	36,691	0.03	7,146,362	256,097	0.03½	8,368,362	292,788
1904	996,456	29,504	0.03	6,391,566	226,481	0.03½	7,388,022	255,985
1905	1,545,400	50,225	0.03¼	6,444,083	219,198	0.03⅓	7,989,483	269,423

The Everett plant of the American Smelting and Refining Company, treating the arsenical pyrite of the Monte Cristo mines of Washington, increased its output. The arsenic deposits on Mineral creek, Lewis county, Washington, were still under development, most of 1905 having been occupied in enlarging and altering the plant, which came into full operation early in 1906. A considerable amount of white arsenic was recovered during experiments, none of which was marketed. The ore here is a massive realgar, from which a very pure arsenious oxide can be made. The plant is modeled after European works and employs no peculiar or patented methods. The engineer for the Mineral Creek Mining and Smelting Company is designing another plant, for the Monte Cristo district, which will be more elaborate, since the ore is more complex, and it is intended to extract the gold as well as the arsenic.

The arsenopyrite mines at Rewald, Va., owned by the United States Arsenic Mines Company, of Pittsburgh, underwent development during 1905, in the search for further ore supplies. In the meantime, production of white arsenic at the smelting plant at Brinton was suspended.

The principal event of 1905 was the beginning of arsenic production at the Washoe smelter, at Anaconda, Mont, belonging to the Amalgamated Copper Company. The arsenic is recovered in dust "kitchens" in the side

flue from the Brunton roasters, and is re-sublimed and crystallized in a specially constructed refinery, built in 1904. The white arsenious oxide, averaging 99.8 per cent. pure, is shipped in carload lots. One new arsenic furnace has been ordered and the capacity of this department will be enlarged.

The ore treated at the Washoe smelter carries from 0.5 to 1, averaging 0.6 per cent., As_2O_3, and the main-flue dust, which is the material that is roasted in the Brunton furnaces, carries from 8 to 15 per cent. As_2O_3. The sublimate from the Bruntons contains from 95 to 98 per cent. As_2O_3. The first roasting takes place in a Brunton furnace and the arsenic content of the residue is reduced to 4 or 5 per cent.; this residue carries non-volatile arsenic obstinately united with iron and calcium, and it is returned to the large reverberatory matte furnaces. The volatile "kitchen" sublimate contains 98 per cent. arsenious oxide, and this is refined to yield the finished product.

Canada.—The production of arsenic in Ontario practically ceased with the closing of the Canadian Goldfields, Ltd., in 1902; the output in 1903 came merely from the working of residues and tailings. The production

WORLD'S PRODUCTION OF ARSENIC.
(In metric tons.)

	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Canada (*a*)...	....	....	....	52	275	630	726	233	66	(*c*)
Germany (*b*)..	2,632	2,987	2,677	2,423	2,414	2,549	2,828	2,768	2,829	2,535
Italy,........	320	200	215	304	120	....	....	....	....	....
Portugal.....	....	524	751	1,083	1,031	527	736	698	....	(*c*)
Spain (*a*)....	271	244	111	101	150	120	71	1,088	400	(*c*)
United K. (*a*)	3,674	4,232	4,241	3,890	4,146	3,416	2,131	916	992	1,552
United S. (*a*).	....	....	....	....	....	272	1,226	554	452	701
Total......	6,897	8,187	7,995	7,853	8,136	7,514	8,718	6,257	4,739	....

(*a*) Arsenious acid. (*b*) Oxide, sulphide, etc. (*c*) Not yet reported.

of 72 tons, valued at $903, in 1904, came entirely from the cobalt and nickel arsenides of Coleman township. Efforts are being made to bring about a resumption of operations at Deloro, where there is still undoubtedly a large amount of arsenical ore.

In the winter of 1903 notable discoveries of arsenic were reported from a point in unsurveyed territory along the line of the Timiskaming & Northern Ontario Railroad, south of Buck township, the arsenic ranging in various assays from 49 to 67 per cent. Within the last two years exploratory work has been carried on at Arsenic Lake, Western Ontario, on the line of the same railroad, two miles north of Temagami. The ore found here is arsenopyrite filling a shear zone, 8 ft. wide, in the green schist of the area. Two grades of ore are here distinguished: one assaying $16.63 gold and silver per ton and 30 per cent. arsenic; the other, 10 per cent. arsenic. It is thought that the number of arsenic deposits found in this immediate area may warrant the erection of an arsenic refinery in their midst. The Arsenical Ore Reducing Company

is actively developing its property at Grey's Siding, two miles north of Temagami. The capacity of the plant is being doubled.

Germany.—This is now the leading source of the world's supply of arsenic. The ore comes from the Reicher Trost mine in the Breslau district, whose annual output is 3500 tons, valued at $3 to $4 per ton. The Samson mine of Clausthal contributes a small quantity of arsenic ore, which, on account of its silver contents, is valued at $170 per ton. The output of arsenic products comes mainly from Freiberg. Germany in 1904 produced 2829 tons of arsenic compounds from 4293 tons of ore.

Italy.—One small mine in the Iglesias mining district now furnishes the entire output of arsenic ore. In 1904 it gave 80 tons of ore carrying 30 per cent. arsenic, and valued at $20 per ton. Another source is the gold-smelting works at Milan, which treats an auriferous arsenopyrite. Imports of arsenic compounds are small, viz., four tons in 1903 and 1904. There is no export.

United Kingdom.—Arsenopyrite is obtained at the tin and copper mines of Cornwall and Devon. Most of it is smelted crudely, at the mines, into "arsenic soot" and sold to the refiners, of whom there are three in Cornwall, two in Devonshire and one at Swansea. One mine ships its crude ore, 47 tons in 1905 and 43 tons in 1904, to the smelter. The output of white arsenic from all the other mines, four in Cornwall and one in Devonshire, was 1528 long tons in 1905 as against 976 tons in 1904. Two decades ago this region was the largest arsenic producer in the world, with an annual output of 8000 long tons of white arsenic. Carn, Brea & Tincroft, and East Pool & Agar are now the two important producers.

ASBESTOS.

The production of asbestos in the United States in 1905 increased to 3100 tons (of 2000 lb.), valued at $126,300, or more than double the output in 1904. Every producer contributed to the increase, but the most noteworthy event of the year was the entrance, in December, of the Hance Asbestos Company, of Grandview, Ariz., into the productive and shipping stage. This is now the only active development of chrysotile asbestos in the United States, the Virginia and Georgia mineral all being amphibole. Chrysotile has been mined in past years in Vermont and Massachusetts. Recent discoveries of this, the most valuable kind of asbestos, have been reported from Arizona and North Carolina. The production of this class of mineral in 1905 accounts for the remarkable increase in the value per ton which is shown in the following table:

ASBESTOS STATISTICS OF THE UNITED STATES.

Year	Production.			Imports		
	Short Tons	Value	Value per Ton	Manufactured	Unmanufactured	Total
1896	716	$12,670	$17.69	$15,654	$229,084	$244,738
1897	840	12,950	15.42	10,570	264,220	274,290
1898	885	13,425	15.17	12,899	287,636	300,535
1899	912	13,860	15.20	8,946	303,119	312,068
1900	1,100	16,500	15.00	24,155	331,796	355,951
1901	747	13,498	18.08	24,741	667,087	691,828
1902	1,010	12,400	12.27	33,313	729,421	762,734
1903 (a)	887	16,760	18.90	32,058	657,269	689,327
1904 (a)	1,480	25,740	17.40	51,290	700,572	751,862
1905	3,100	126,300	40.74	70,117	776,362	846.479

(a) Statistics of the United States Geological Survey.

The Hance asbestos mines are situated at the bottom of the Grand Cañon of the Colorado, on the north side of the river, 5000 ft. below the rim of the cañon. Grandview, on the south rim, is the nearest town; this is 16 miles by wagon road from the nearest railroad point, which is at the end of a branch from the Santa Fe railroad at Williams. In the cañon, a basaltic dike, 30 to 70 ft. thick, has intruded between the Archean crystalline rocks and the overlying Algonkian sedimentaries. Wherever the dike has come in contact with limestone, it has itself been altered to serpentine, and the chrysotile is contained in this. The serpentine bands in places hold a constant width, averaging 15 in., for distances of 150 ft. or more, contrary to the usual habit of serpentine. Quarrying and drifting have opened up the deposit. The asbestos seams range from $\frac{3}{4}$ to $1\frac{1}{2}$ in. in width. The mineral is of a transparent golden color when in masses, but white and silky when fiberized. The

mechanical development of the mine will be difficult but not impossible; an aerial tramway is designed to cross the cañon at a height of 1200 ft. above the river, to facilitate shipping the product.

Chrysotile asbestos is in growing demand for the manufacture of cloth and cordage, where its superior tensile strength makes it indispensable. The amphibole variety, though occurring in much longer fibers, up to two feet, and in larger and more compact masses, so that its mining is cheapened, cannot be used alone for spinning, but is limited to the manufacture of board, molded articles, and lagging for fireproofing and insulating purposes. The average price of amphibole asbestos is $12.50 per 2000 lb., while crude chrysotile sells for $100 or over. The consumption of chrysotile in the United States is supplied by importations from Canada, which amount to more than 20 times the value of the domestic production.

The United States, Canada and Russia supply practically the world's entire consumption of asbestos. South Africa and Italy afford other supplies in small but unascertainable quantities. The productions of these three countries are grouped in the following table:

ASBESTOS PRODUCTION OF THE WORLD.

(In metric tons.)

	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Canada	9,883	11,980	14,631	16,143	19,619	29,847	27,422	28,248	32,337	45,980
Russia	1,131	1,274	1,665	2,693	*e*3,000	4,399	4,507	*a*	*a*	*a*
United States	650	762	803	828	998	678	917	805	1,343	2,813
Total	11,664	14,016	17,099	19,664	23,617	34,924	32,846			

(*a*) Not yet reported. (*e*) Estimated.

Prof. George F. Seaver, of Columbia University, New York, has made a series of tests on asbestos and on magnesia building lumber, to determine their relative durabilities and other properties. The material tested was in ⅛-, ¼- and ½-in. thicknesses. It was supplied by the Keasbey & Mattison Company, of New York, for whom the tests were made. The first and second tests, on electric resistance and breakdown voltage, showed the magnesia board to rate higher than the asbestos. Test No. 3, in which the specimens were subjected to the heat of a 500-kw. arc lamp for 20 seconds, demonstrated that asbestos board has a better fire or arc resistance than magnesia board. Test No. 4, on heat conductivity, showed that asbestos board is a much better non-conductor of heat than magnesia board. Test No. 5, for vibratory strength, demonstrated that asbestos board withstands heavy blows and hard usage better than magnesia board. Test No. 6, for determining the absorbent characteristics of the two materials, showed that, while the asbestos board was not absolutely moisture-proof, still the percentage of moisture absorbed was small and had no deteriorating effect upon the board. Test No. 7, as to the ease with

which these boards could be sawed and nailed, showed that the magnesia board could be more readily worked with tools than the asbestos, but was more liable to split.

Professor Seaver concluded that asbestos board possesses better mechanical properties than the magnesia, but that the magnesia is a better electric insulator than the asbestos. The latter, however, stands better as a heat-resisting and arc-resisting medium than the magnesia, and thus, from the standard of protection against fire, asbestos board appears to possess more merit than magnesia board.

Other tests were made upon asbestos mill-boards, $\frac{1}{8}$-, $\frac{1}{4}$- and $\frac{1}{2}$-in. thick, by Prof. Woolson in the testing laboratory of Columbia University. The tests consisted in exposing samples to different high and long-continued heats; the results demonstrated that a high heat causes asbestos fibers to become brittle, though without destroying their heat-resisting properties.

The various asbestos boards on the market are said to be composed of 25 per cent. asbestos and 75 per cent. portland cement, thoroughly mixed and submitted in sheets to a pressure of about 85 lb. per square foot. The magnesia board or sheathing is prepared by permeating the asbestos mill-board with a solution of sodium silicate and magnesium bicarbonate, the water being removed by pressure. Such lumber is employed in the construction of street railway and electric railroad cars, for protecting them from fire by any failure in the electric apparatus. Among others, the New York City Railroad Company is now using asbestos for the lower boards of new double-truck cars which it has recently built. The Interborough Rapid Transit Company, the General Electric Company and the Brooklyn Rapid Transit Company are also using asbestos board for various purposes. The Montreal Street Railroad Company has also recently specified that this material shall be used in cars now being built for its use.

Asbestos in Canada[1].

Mining for asbestos in Thetford and Coleraine townships of Quebec began in 1878, but never acquired importance until the installation of mechanical means for dressing the mine product within the last few years enabled the producers both to mine a lower grade of rock and to save a larger proportion of the valuable portions. Sixteen mills are now operating in the district, having a united daily capacity of 3500 tons of crude rock.

Except in one locality, near Elzevir, in Hastings county, where actinolite is mined and dressed, the whole Canadian output is of the chrysotile variety. It occurs to some extent in the Laurentian serpentinous limestone around Templeton, to the north of Ottawa, but the productive centers are confined to the serpentine areas in the eastern townships of Quebec, two

[1] Abstract from "Asbestos, its Occurrence, Exploitation and Uses," by Fritz Cirkel, published by the Mines Branch of the Canadian Department of the Interior, 1905.

distinct fields—Thetford-Black Lake and Danville-Oxford Patton—being recognized, with another in East Broughton.

The serpentine rock, carrying veins and stringers of chrysotile, is mined universally in open quarries. Attempts to use underground methods, in the hope of escaping delays during winter weather, have been unsuccessful, owing to the erratic nature of the veins, the interference by wide, barren zones and the large proportion of rock necessary to be moved. One mine, however, in exceptionally favorable circumstances has accomplished this method of operation. The greatest difficulty to contend with in quarry work is the disposal of waste; early operations have in many cases covered valuable ground with waste dumps.

Surface soil, varying in depth from 2 or 3 ft. at Black Lake to 15 or 20 ft. at Thetford, is removed by pick and shovel and loaded into dumping cars on movable tracks. One Thetford company has installed a steam shovel for this work. For drilling on the benches, steam or air machine drills are common, Ingersoll and Rand, $3\frac{1}{8}$x$6\frac{3}{4}$-inch size, being the most popular. Block holing is done by hand and by Little Giant drills. The common explosive is 40 per cent. dynamite, and electric firing is usual. The average breakage is $4\frac{1}{4}$ to 5 tons of rock per pound of explosive. A few quarries have passed 150 ft. in depth. Rock is hoisted by boom derricks or by suspension cable derricks; the boxes in which it is handled hold between 16 and 20 cu. ft. of rock, weighing 2200 to 2500 ℔. Dumping cars for transport to mill, at the large mines, hold 3 to 6 tons, and are handled by locomotives.

The amount of hand dressing devoted to the product in the quarry varies with circumstances. Some mines send the entire output to the mill; others make a rough separation into:

(*a*) "Crude," the long asbestos fiber and the rock containing it.

(*b*) "Milling material," the rock containing shorter fiber.

(*c*) "Fine material," resulting from fracture of the rock.

(*b*) "Waste."

The crude is reserved for hand cobbing; the second and third classes are sent to mill, and the waste is discarded.

The crude is separated in cobbing sheds, generally into two grades, No. 1 measuring over $\frac{3}{4}$ in. and No. 2 exceeding $\frac{5}{16}$. This hand work is commonly done by contract at 30 to 35c. per 100 lb.; the cost ranges from \$6 to \$8 per short ton.

The only pieces of apparatus peculiar to asbestos mills are beaters or "fiberizers," suction elevators and collectors; all other drying, crushing, grinding, and screening machines are of the well-known standard types. One kind of fiberizer consists of a horizontal stationary cylinder, 30 in. diameter and 12 ft. long, through the axis of which passes a shaft carrying groups of four perpendicular arms, the groups separated by 6 in. Th

shaft is revolved 500 to 700 r. p. m.; asbestos-bearing material is fed into both ends at the top, and is discharged at the middle of the bottom, the beating blades being set to propel the material toward the center. The cyclone fiberizer consists of a pair of chilled steel propellers facing each other near the bottom of a cast-iron chamber, and revolving at 2000 to 2500 r. p. m. in opposite directions. A propeller wears about two weeks. The feed is about walnut size and the discharge about pea size; the capacity on rock of average hardness is about 40 tons per hour.

The suction elevator, the motive power of which is a fan of 30 to 40 in. diameter driven at 2000 r. p. m., terminates in a cone flattened so as to span the lower end of a shaking screen, presenting a rectangular opening 6 in. wide, into which the loose fibers are drawn. The collector is a closed conical hopper into which the fiber-laden blast from the suction fans enters tangentially near the top; dust escapes through an opening in the center of the cover, while the fiber falls out at the bottom.

The method of milling differs in every mill. In general the rock first passes through a jaw crusher and then through a rotary drier, whence it is elevated to a rotary crusher at the top of the mill. After this, it is further reduced in the cylindrical fiberizer and then passed over a shaking screen with $\frac{1}{16}$ in. holes, the loosened fiber being lifted by suction from the lower end to be collected as first grade. All the remaining material passes successively through cyclone beaters, grading screens and grinders until the maximum amount of fiber is recovered. Tailing is pulverized for use in plaster.

Besides the two grades of "crude" obtained by hand cobbing, usually three grades of product are obtained in the mill, designated as "long spinning fiber," "spinning fiber," and "paper stock." One mill recovers these in proportions of 15.6, 32.2 and 53.2, respectively. As to the total yield from quarried rock, the general proportion is 2 per cent. of crude, and between the limits 30 per cent. and 60 per cent. of milling rock; the latter yields between 6 and 10 per cent. of fiber in the mills. The cost of labor at one mill, capacity 250 tons of rock per day, is $0.25 per ton of mill rock, or $2.56 per ton of finished fiber. For the whole operation, on the basis of an average output of 7.5 tons of asbestos fiber per day, the costs at one mine for three consecutive months were $8.86 for mining and $8.55 for milling, total $17.41 per ton of finished product. Of this total, labor amounted to $10.13; power, $4.19; bags, $1.40; machinery, repairs and supplies, $1.01; explosives and other supplies, $0.68.

Current prices obtained per ton for the various grades are as follows: No. 1 crude, $175 to $200; No. 2 crude, $110 to $125; fiber No. 1, $75 to $80; fiber No. 2, $50; paper stock, $20 to $25.

There are now 14 incorporated companies, of which 10 are operating at this time. Four companies are in the Thetford district, viz., the Johnson,

the Bell, the King Bros. and the Beaver asbestos companies. Seven are at Black Lake, viz., the Standard, the Manhattan, the Glasgow and Montreal, the Union, the Johnson, the Syracuse and the American; only two

STATISTICS OF ASBESTOS IN CANADA. (*a*)

(In tons of 2000 pounds.)

Year (*b*)	Production				Exports (*c*)		Imports (*d*)
	Asbestos		Asbestic				
	Short Tons.	Value	Short Tons.	Value.	Short Tons.	Value	Value
1896	10,892	$ 423,066	1,358	$6,790	9,588	$482,679	$ 23,900
1897	13,202	399,528	17,240	45,840	10,969	510,916	19,032
1898	16,124	475,131	7,661	16,066	18,424	510,368	26,389
1899	17,790	468,635	7,746	17,214	14,520	453,176	32,607
1900	21,621	729,886	7,520	18,545	18,164	490,900	43,455
1901	32,892	1,248,645	7,325	11,114	26,715	864,573	50,829
1902	30,219	1,126,688	10,197	21,631	33,072	1,131,202	52,464
1903	31,129	915,888	10,548	13,869	30,661	955,405	75,465
1904	35,635	1,167,238	13,011	13,006	34,636	984,836	83,827
1905	50,670	1,486,359	17,594	16,900	41,127	1,311,524	116,836

(*a*) From *Annual Reports* of the Geological Survey of Canada, and the *Statistical Year Book* of Canada. (*b*) Production is given for calendar year; exports and imports are for fiscal years ending June 30. (*c*) Mainly crude asbestos. (*d*) Manufactured articles entirely.

are now working at East Broughton, these being the Quebec and the East Broughton; while all the mines at Danville are now owned by the Asbestos & Asbestic Company, Ltd.

Uses of Asbestos.

The principal difficulty in the way of spinning asbestos is its lack of cohesiveness in a strand, owing to the absence of the roughness in the fiber that characterizes animal and vegetable tissue. Spinners have succeeded, however, in producing an asbestos thread of fair strength weighing 1 oz. per 100 yd. One of the most satisfactory uses of asbestos is as piston-rod packing in compound engines using superheated steam. Woven into cloth, or twisted into ropes, asbestos is useful for its fire-resisting property. As a heat-insulating material it is applied to boilers and pipes either as a plastic, or in molded casings. Its most widespread use is in fireproofing and building materials, of which numerous patented forms under a variety of names are now on the market.

ASPHALTUM.

THE production of asphaltic materials of all kinds in the United States, so far as statistics are available, is reported in the following table:

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES.
(Tons of 2,000 lb.)

States.	1903. (a)			1904. (a)			1905.		
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.
Bituminous Sandstone......									
California...............	24,080	$69,862	$2.90	6,814	$19,264	$2.83	22,500	$63,000	$2.80
Kentucky...............	11,628	38,763	3.33	5,670	31,185	5.50	7,530	34,885	4.63
Indian Territory.........	1,710	3,908	2,29	5,457	12,516	2.30	(e)5,000	11,500	2.30
Arkansas...............	1,215	5,468	4.50	1,700	8,500	5.00	(e)1,500	7,500	5.00
Total.................	38,633	$118,001	$3.05	19,641	$71,465	$3.64	36,530	$116,885	$3.20
Asphaltic Limestone........	2,520	8,800	3.49	1,798	$4,495	2.50	2,000	5,000	2.50
Asphaltum (b)....									
California...............	50,487	632,764	12.53	53,465	626,165	11.71	57,687	545,503	9.46
Indian Territory(c)......	877	15,442	17.61	*Nil.*	......	.....	*Nil.*	....	....
Missouri...............	.....	......	.....	937	16,676	17.80	(e)1,000	17,500	17.50
Texas..................	2,158	30,550	14.16	25,000	250,000	10.00	113,500	851,250	7.50
Total Asphaltum.......	53,522	$678,756	$12.68	79,402	$892,841	$11.24	172,187	$1414253	$8.20
Gilsonite (d)									
Utah..................	5,619	188,357	33.52	3,528	155,080	43.93	10,516	110,144	10.47
Indian Territory(c)......	.....	......	.....	1,000	25,000	25.00	(e)1,000	25,000	25.00
Mastic:									
California...............	11	132	12.00	*Nil.*	....	....	*Nil.*	....	....
Kentucky...............	950	11,400	12.00	*Nil.*	....	....	*Nil.*	....	....
Pennsylvania............	.....	......	.....	1,200	10,800	9.00	2,000	18,000	9.00

(a) From the Mineral Resources of the United States. (b) Includes hard and refined, or gum, liquid or maltha, and oil residues. (c) Includes production of Oklahoma Territory. (d) Includes gilsonite, elaterite and grahamite. (e) Estimated.

Colorado.—The gilsonite veins of Utah are said to enter the western part of Rio Blanco county, Colo., and considerable product has been mined and shipped. Elaterite exists near Rangeley, three veins, 4 in. wide, having lately been discovered on Piceance creek. Near by are deposits of sand asphaltum.

Utah.—The Uteland Mining Company, with a capital of $4,000,000, expects to operate in the Utah reservation, where it has acquired 213 acres of hydrocarbon lands once owned by the Florence Mining Company.

Statistics of most of the asphalt-producing countries are late in arrival; we have, however, secured the following statistics of asphalt production in 1905: Germany, 103,006 metric tons; Trinidad, 116,735 metric tons; Venezuela, estimated, 25,000 tons.

Cuba.—The Mariel mine in Pinar del Rio province, owned by the Cuba Minera Industrial, is now the only active shipper of Cuban asphalt, all other producers, among them the Cuban American and the West Indies companies, having succumbed to the competition of temporarily low-priced Venezuelan asphalt. The Talaren mine, Puerto Principe, which yields

only glance pitch, has been crowded out of the market by the cheaper Utah gilsonite. The plant of the West Indies Company's Angela Elmira mine is still on the ground, ready to be started as soon as the market improves.

Trinidad.—In addition to the resources of Pitch Lake and contiguous land, deposits of "manjak" are being developed on a scale corresponding to the growing demand for the substance. The Trinidad manjak deposits

WORLD'S PRODUCTION OF ASPHALT AND BITUMINOUS ROCK.
(In metric tons.)

Asphalt.

Year.	Cuba.	Germany.	Hungary.	Italy.(*c*)	Russia.	Spain.	Trinidad (*b*)	United States.	Venezuela.	Total.
1900.....	(*d*)	89,685	2,900	33,127	25,090	2,331	161,299	8,326	17,981	340,739
1901.....	4,554	90,193	2,878	31,814	26,622	4,182	173,707	19,882	22,115	375,947
1902.....	4,966	88.374	2,773	33,684	12.360	6,064	167,253	36,923	10,770	363,167
1903.....	7,368	87,454	2,422	32,000	(*d*)	4,372	196,883	54,521	14,567	399,587
1904.....	8,926	91,736	2,221	31,327	(*d*)	3,761	137,089	77,250	23,535	375,845

Bituminous Rock.

Year.	Austria.	France.(*a*)	Italy.	Spain.	United States.	Total.
1900..........	887	34,093	101,738	4,193	41,029	181,940
1901..........	541	29,815	104,111	3,956	37,393	175,816
1902..........	901	34,000	64,245	6,301	35,072	140,519
1903..........	1,273	(*d*)	89,690	6,277	37,334	134,574
1904..........	1,435	(*d*)	111,900	100	19,454	132,889

(*a*) France produces a large amount of bituminous shales, used for distilling oil, which is not included in these statistics. (*b*) Exports (crude equivalent) reported by the New Trinidad Lake Asphalt Co. (*c*) Including mastic and bitumen. (*d*) Not yet reported.

Note.—There is a considerable production of asphalt stone in Switzerland of which no account is taken in the above table, the Swiss Government not publishing any mineral statistics. The production of manjak in Barbados is not included in the statistics given.

were discovered several years ago on the abandoned Vistabella sugar plantation, one mile from the town of San Fernando, and 12 miles from Pitch Lake, with which they have no apparent connection. The Vistabella estate, covering 360 acres, lies on the west sea-coast just south of the Tarouba river, and is traversed by the Government railway. The deposits were worked by a private syndicate until July, 1905, when the Trinidad Vistabella Manjak Company, Ltd., Port-of-Spain, Trinidad, was organized.

Thus far, of the several deposits on the estate, only one has been productive, and this has supplied the entire demand. Its total output has now reached 6000 long tons; the yield in 1904 was 3000 tons, but a serious accident early in 1905, in which 17 lives were lost, restricted the output of that year. More recently, an output of 500 tons per month has been sustained for several months. It is anticipated that the demand will continue to grow, as the possibility of maintaining a steady production becomes assured. A large firm in London has already signified some desire to contract for the entire output.

The manjak occurs in a steeply pitching seam, which widens with depth. A vertical shaft has been sunk to cut the seam, and crosscuts have been

driven on five levels to intersect it. On the first level, 55 ft. below surface, the seam is 10 ft. wide, and on the fifth, at a depth of 200 ft., it is 33 ft. wide. Down to the fourth level, at a depth of 150 ft., the manjak retained a dull coaly character; on the fifth level, the seam was found to have a core of bright, unaltered material with the shiny gloss of patent leather.

Of the following three analyses of Vistabella manjak, the first two are by Boverton Redwood, and their percentages are based on the dried material; the third is by P. Carmody, and is based on the original sample:

	I.	II.	III.
Water	4.2	4.4	0.4
Petrolene	17.5	20.3	16.8
Asphaltene	71.2	66.9	72.0
Total bitumen	88.7	87.2	88.8
Non-bituminous organic matter	5.0	6.5	0.4
Mineral matter	6.3	6.3	4.2
Sulphur	2.97	3.06	2.6
At red heat:			
Coke	35.7	36.7	30.65
Volatile	58.0	56.9	65.15
Specific gravity	1.139	1.147	1.127
Softening point	300° F.	300° F.	383° F.
Melting point	360° F.	360° F.	464° F.

Another analysis by the New York Testing Laboratory shows 94.2 per cent. of bitumen soluble in carbon bisulphide, and three more by A. Urich show soluble bitumen ranging from 95 to 97.6 per cent.

EXPORTS OF ASPHALT FROM TRINIDAD.
(In tons of 2240 lb.)

Year.	Pitch Lake Asphalt.		Land Asphalt.		Total.	
	To United States.	Elsewhere.	To United States.	Elsewhere.	To United States.	Elsewhere.
1900 (a)	70,938	51,805	34,796	448	105,734	52,253
1901	80,449	55,605	31,767	3,150	112,216	58,755
1902	104,956	34,220	25,153	290	130,109	34,510
1903 (b)	123,582	41,950	18,478	4,886	142,060	46,836
1904 (c)	63,033	48,655	22,582	660	85,615	49,315
1905 (c)	47,947	54,054	12,126	770	60,073	54,824

Note.—A small proportion of the output undergoes a slight refining process before shipment. The above figures give the total exports, after recalculating, the "épuré and dried" to their equivalent original amounts of crude material. Shipments elsewhere than to the United States are mainly to Europe. (a) Epuré and dried are not reduced to original crude in 1900. (b) For thirteen months ending January 31, 1904. (c) Years ending January 31, 1905, and 1906.

Venezuela.—Asphalt from Venezuela within two years has been the disturbing element in the market. The New York and Bermudez Company having defrayed all the expense of developing the deposits at Bermudez Lake, it follows that the receiver appointed by the Venezuelan court, after the rightful owner had been dispossessed, is able to sell his product at a price lower than is possible with companies operating elsewhere. This has forced severe competition, particularly upon the Trinidad operators.

Asphalt Paving Industry.

The General Asphalt Company is now the most important factor in the asphalt paving industry of the United States. Among other operators

in the same line are the Warner-Quinlan Company of Syracuse and the Uvalde Asphalt and Manufacturing Company, the Sicilian Asphalt Paving Company, and the A. L. Barber Asphalt Company, all of New York City. The asphalt industry as a whole has suffered by extravagance and loose financiering and, in its connection with the Bermudez deposits of Venezuela, has led to a vast amount of litigation, assuming at times the proportions of an international controversy.

The General Asphalt Company is a holding corporation, owning and directing three subsidiaries, the Barber Asphalt Paving Company (not the A. L. Barber company above mentioned), which in turn controls numerous smaller paving companies, including now the South American Asphalt Paving Company, which until 1905 constituted the fourth of the main company's subsidiaries, the new Trinidad Lake Asphalt Company, Ltd., and the New York & Bermudez Company. The first is engaged exclusively in paving; the second is occupied in mining at Pitch Lake, Trinidad; and the third was, until the seizure of its property in July, 1904, engaged in similar work at Bermudez Lake, Venezuela, since which time its principal occupation has been to resist the quasi-legal encroachments of its rivals in that field and the annihilation of its rights by the Venezuelan Government.

The Barber Asphalt Paving Company, during the year ending Jan. 31, 1906, paved, with asphalt and asphalt blocks, 3,860,811 sq. yd., of which about two-thirds was done for municipalities, as compared with 3,103,381 sq. yd. in 1904. Contracts unperformed at the end of the year called for an additional 743,759 sq. yd. The company manufactured 61,578 tons of refined asphalt during 1905. The combined operating accounts of all the general asphalt companies' subsidiaries for the year ending Jan. 31, 1906, may be shown thus:

Cost of crude asphalt	$304,43	Sales of crude asphalt	$471,055
Cost of refining asphalt	2,080,395	Increased inventory value	23,491
Cost of miscellaneous materials	546,015	Sales of refined asphalt	2,905,046
Cost of paving	6,404,666	Sales of miscellaneous materials	523,117
Maintenance and repair of paving	250,221	Income from paving	6,448,378
Cost of private work	1,387,080	Income from private work	1,764,267
Depreciation of plants	149,243	Income from contracting and miscellaneous work	1,090,265
Contracting and miscellaneous work	1,062,156	Miscellaneous income	97,729
Sundry branch expenses	22,759		
Balance of profit	1,116,679		
	$13,323,348		$13,323,348

The net profit on the year's operations, after deducting office and sundry general expenses, and interest, was $532,991, out of which a dividend of $262,793, at the rate of 2 per cent. on the preferred stock, was paid.

BARYTES.

THE production of barytes in the United States in 1905, compiled from reports received by us from the producers and railway companies, was 53,252 tons of 2000 lb., valued at $196,041, to be compared with 65,727 tons reported by the United States Geological Survey as the output in 1904. The output in 1905 was contributed by Missouri, Tennessee, North Carolina and Virginia, named in order of importance.

STATISTICS OF BARYTES IN THE UNITED STATES.

(In tons of 2000 lb.)

Year	Production			Imports				Consumption.	
	Short Tons	Value		Crude		Manufactured			
		Per Ton	Total	Sh. Tons	Value	Sh. Tons	Value	Sh. Tons	Value
1896	21,900	$4.00	$ 87,600	509	$ 1,274	2,467	$23,345	24,876	$112,219
1897	26,430	4.00	105,720	502	579	1,300	13,822	28,232	120,121
1898	28,247	4.00	112,988	1,022	2,678	687	8,678	29,956	124,344
1899	32,636	4.20	137,071	1,739	5,488	2,111	22,919	36,486	165,478
1900	41,466	3.90	161,717	2,568	8,301	2,454	24,160	46,488	194,178
1901	49,070	3.22	157.844	3,150	12,380	2,454	27,062	54,674	197,286
1902	58,149	3.21	186,713	3,929	14,322	3,908	37,389	65,986	238,424
1903	a50,397	3.02	152.150	6,344	22,777	5,716	48,726	62,457	223,653
1904	a65,727	2.66	174,958	6,689	27,463	5,920	48,658	78,336	251,070
1905	53,252	3.68	196,041	7,879	36,796	4,287	39,803	65,418	272,649

(a) Statistics of the U. S. Geological Survey.

At present, only two districts supply barytes in significant quantities, and these in about equal amounts. The Missouri district covers Washington, St. François, Crawford, Cole and Miller counties. Washington county produces 75 per cent. of the output of Missouri. The production of the State in 1904 (according to the mine inspector) was 44,161 tons, valued at $176,644, or $4 per ton. Our reports show a production of 27,432 tons in 1905. More attention is now given to the mining of barytes in Washington county than of lead. The demand for the mineral has increased, and it is not only far more abundant than the lead ore of the district, but it is more easily found and mined.

The Appalachian district covers both sides of the Cumberland mountains and Blue Ridge, including contiguous portions of Virginia, Tennessee, and North Carolina.

The Missouri output is obtained from widely scattered localities, most of the workings being on a small scale, and is shipped chiefly to St. Louis for treatment. But a mill erected in 1904 by the Point Mining and Milling Company, at Mineral Point (near Potosi), Washington county, Mo., now affords a local market. This mill has capacity for grinding 30 tons of barytes per day, and is claimed to be the best equipped in the country.

The centers for treating the Appalachian product are Bristol and Sweetwater, Tenn. The value of crude barytes at the mines ranges from $3.50 to $4 per short ton.

Virginia.—In connection with some of the barytes deposits of Virginia, the problem is not one of mining, but rather of handling the material by industrial railways and other similar appliances. However, the deposits are so scattered and so small that the proper appliance to handle the dirt and ore to the best advantage would clean out some deposits in a few weeks.

The best and most available barytes occurs in a sort of loamy dirt. An ideal way to handle this would be with a steam shovel and log washer, somewhat as brown ore is handled in the South, but as a thousand tons of barytes from one lease is an exceptionally large amount, it would be impractical to install such a plant. The usual method is to make an open cut, follow the barytes where it leads and reject the dirt. Below the dirt the barytes runs into the limestone, and if cheap transportation is available it pays to work that. In some places, far from the railroad, the cost of teaming is greater than the cost of mining. The best and most abundant deposits seem to be the farthest from the railroad.

The limestone deposits might be worked more easily by the use of jigs. The uncertainty of the business, however, has prevented such innovations. In the Virginia district it is hard to accumulate sufficient barytes to run a good-sized mill. The soft nature of the barytes is rather against mechanical handling, and even if there were a sufficient amount to warrant putting in a steam shovel and washer, the waste might be too great.

Barytes Mining in Foreign Countries.

Canada.—The Lake Ainslie Mining and Railway Company, Ltd., formerly the Canadian Barytes Company, Ltd., is actively developing its barytes property at East Lake Ainslie, Inverness county, Cape Breton Island.[1] The company has already spent $70,000 in opening its mine and in equipping its three grinding and refining works at Dartmouth, Nova Scotia, in Brooklyn, N.Y., and at Montreal, Quebec. The first two have been in operation for some time, and the third will soon begin.

The barytes veins cut diagonally, northeast to southwest, and dip to the south, through a 500-ft. hill on the east shore of Lake Ainslie. Four veins have been opened; the southern one, which outcrops on the lake side of the hill, is 12 to 14 ft. thick, between well-defined walls. The other outcrops on the land side of the hill, one of them being 16 ft. thick and the other two each 8 ft. thick and separated by 20 ft. of rock.

The veins have been explored on both sides of the hill, but the main output is now coming from open work on the land side, whence a series of trams and chutes delivers the product to pockets on a 300-ft. wharf at the lake.

[1] J. W. Regan, *Industrial Advocate*, Halifax, N. S., September, 1905, p. 19.

A steam lighter of 50 tons capacity then transports the barytes across the lake eight miles to Strathlorn, where it is delivered to the Inverness Railway and Coal Company, which hauls it to Port Hastings, 58 miles, whence the product is exported to England and the United States.

Mining is being carried out systematically with a force of 45 men. Air drills are used in the headings and the drifts are timbered. Mineral positively developed is estimated at 400,000 tons above water level, all of which can be mined without hoisting or pumping. The company plans an output of 7000 tons per year, but is studying the British market with a view to increasing the production, aided by the cheap ocean freights from Halifax.

Technology of Barytes.

A Barytes Plant.—The new plant of the W. D. Gilman Company, at Sweetwater, Tenn., consists of a building 60x190 ft., and 40 ft. high. A side track of the Southern railway runs to the building and is equipped with a weigh-bridge and elevators for loading the clean barytes and barium sulphide into the cars. At the south end of the building is a crusher and grinding-room, where the barytes and coal are prepared for the furnaces. From this room the mixture is carried by conveyors to storage bins over the furnaces, whence it is charged as required. The furnaces are of the usual revolving type, and have a capacity of 20 tons per day. The product is discharged automatically, while still red hot, into iron cars, which are hauled away to the storage room, where they are dumped. When cool, the product is loaded into cars on the side track for shipment. This furnace product contains 65 to 70 per cent. of barium sulphide, and 15 to 20 per cent. of barium carbonate, the residue consisting of unconsumed coal, ash and silica. The conversion of barium sulphate is almost complete. This crude sulphide finds a ready market for the manufacture of lithophone and the chemical salts of barium, of which chloride and nitrate are most in demand. The cheapness with which the ore is mined, and coal being obtained in the district, make it possible to manufacture the sulphide and deliver it in New York at a lower price than it can be made in that city.

The department for cleaning and bleaching mineral for the production of ground bartyes contains a large Blake crusher. From this the mineral goes to a table where it is passed over a magnet to remove iron, and thence to a washer which frees it from clay. From the washer it goes to the rougher and cleaner, after which it is ready for the bleaching tanks, where it is treated with acid and steam in the usual way. The output is 30 tons per day.

Barytes in the Sweetwater district seems especially adapted to grinding, as it is very pure and soft. So far, it has improved in whiteness as the

mines go down. The greatest depth yet attained by any working in this field is about 100 ft.

Process for Making Barium Carbonate.—A process for making precipitated barium carbonate from barytes, described by P. Seurre, consists in heating, to the fusion point, equal parts of pulverized barytes and calcium chloride, with five or six per cent. of powdered charcoal, in a crucible, or a furnace, with constant stirring. The resulting mass, leached in water, yields a solution of barium chloride and an insoluble residue of calcium sulphate. To the filtered chloride solution is added a 50 per cent. solution of ammonium carbonate, when the precipitated barium carbonate is filtered and washed. The ammonium chloride in solution may then be treated, in a still, with chalk, recovering both ammonium carbonate and calcium chloride, both of which can be returned to the process.

BAUXITE.

THE production of bauxite in the United States in 1905 was 47,991 long tons, valued at $203,960, a decrease of only 21 tons from the output in 1904. Its value per ton, however, showed an advance of $0.79.

STATISTICS OF BAUXITE IN THE UNITED STATES.

Year.	Production.			Imports.		Exports.		Consumption.	
	Long Tons.	Value.	Per Ton.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1896	17,096	$ 42,740	$2.50	2,119	$10,477	....	....	19,215	$ 53,217
1897	20,590	51,475	2.50	2,645	10,515	2,537	$5,074	20,708	56,916
1898	26,791	66,978	2.50	1,201	4,238	1,000	2,000	26,992	69,216
1899	36,813	101,235	2.75	6,666	23,768	2,030	4,567	41,449	12,436
1900	23,445	85,922	3.66	8,656	32,968	1,000	3,000	31,101	115,889
1901	(a)18,905	97,914	4.23	18,313	66,107	1,000	3,000	36,218	144,021
1902	29,222	128,206	4.39	15,790	54,410	*Nil.*	....	43,112	175,875
1903	(a)48,087	171,306	3.56	14,889	49,684	*Nil.*	....	62,976	220,990
1904	48,012	166,121	3.46	15,475	49,577	*Nil.*	....	63,487	215,698
1905	47,991	203,960	4.25	11,726	46,517	*Nil.*	....	59,717	250,477

(a) Statistics of the United States Geological Survey.

PRODUCTION OF BAUXITE IN THE UNITED STATES.

(In tons of 2240 lb.)

State.	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Alabama	9,796	13,083	13,848	14,144	650	(a) 18,038	5,577	(a) 22,374	7,087	17,094
Georgia	7,300	7,507	12,943	19,619	20,715		19,000		16,909	
Arkansas	....	....	....	3,050	2,080	867	4,645	25,713	24,016	30,897
Total	17,096	20,590	26,791	36,813	23,445	18,905	29,222	48,087	48,012	47,991

(a) Statistics of the United States Geological Survey.

The most important event of the year was the absorption of the General Bauxite Company by the Pittsburgh Reduction Company. The General Bauxite Company operated in both Arkansas and Georgia, and next to the Pittsburgh Reduction Company was the heaviest producer in the country. The Pittsburgh company confined its mining to Saline county, Arkansas, where it obtained the raw material for its alumina purifying works at East St. Louis; it had, however, other sources of refined alumina. The entrance of the Pittsburgh company into the southern bauxite field, together with its acquisition of additional resources in Arkansas, was doubtless a precautionary step to fortify, by monopolizing the sources of raw material, its position after the expiration of its aluminum patents. The policy is in line with that of other metallurgical undertakings, which find it advisable to control the supplies of raw material. The Pittsburgh Reduction Company has secured a bauxite deposit also, near the railroad, in Shasta county, California; several deposits of the mineral are known to exist in that State.

The southern bauxites have not been used in making aluminum, partly because of the long haul to the alumina purifying works, and partly because

their superior freedom from iron renders them indispensable to the manufacturers of alum and aluminum sulphate.

Bauxite in Foreign Countries.

India.[1]—Burma and the Indian Peninsula contain large undeveloped deposits of bauxite, some of which (as was the case in the early history of the French localities) have been looked upon as a possible source of iron. In general, the Indian bauxites can become profitable only by refining and calcining them on the spot; the current low prices would exclude the crude material, with its high land and ocean transportation charges, from the market. Much of the Indian bauxite, however, contains alumina in excess of the average ore now marketed.

The following are some representative analyses of Indian bauxites:

	I	II	III	IV	V	VI	VII	VIII
Al_2O_3	62.80	45.00	52.67	54.20	57.50	35.38	64.64	58.23
Fe_2O_3	0.44	9.30	7.04	4.02	6.53	34.37	6.21	5.48
H_2O	33.74	28.30	29.83	27.93	26.94	19.00	24.00	28.10
SiO_2	2.78	14.58	1.26	1.55	2.35	10.75	1.79	2.01
TiO_2	0.04	0.52	7.51	12.21	6.61	0.10	3.30	6.49
CaO	0.20	....	1.75	....	0.15	0.40	0.04	0.45
MgO	0.03	1.31	trace	....	*Nil.*	....	0.02	....
	100.03	99.97	100.06	99.91	100.08	100,00	100.00	100.76
Sp. grav	2.42	....	....	...	2.39	3.07	2.59	....

I. From Kodaikanal, Palni hills, Madras. Cream-colored nodules, forming a layer 1 ft. thick, mixed with clay and other decomposition products from the charnockite series.
II. Same locality as No. I.
III. From Katni, Jubbulpore district, central India. Brown pisolite.
IV. From Samnapur, Balaghat district, central India. Yellowish-brown pisolite.
V. From Bijeeragogarh, Marwara, Central Provinces.
VI. From the Nilgiri hills, Madras. Of the SiO_2 reported, 10.52 per cent. was quartz.
VII. From Neturhat, Palamau, Chota Nagpur. Pisolitic.
VIII. From Sarguja State, Chota Nagpur.

To refine this bauxite by the common method would require caustic soda, which is not now made in India. Both caustic soda and bleaching powder are imported largely for use in paper manufacture, and this would suggest the profitable introduction of the electrolytic process for making caustic soda and chlorine from salt.

Bauxite Brick.

The so-called bauxite brick, made simply by mixing coarsely ground bauxite with fireclay, has done much to bring bauxite refractory products into disrepute.[2] The purest fireclays have a melting point of 1850 deg. C. and manufactured firebrick has as highest limit the melting point 1830 deg. C. The melting points of bauxite differ according to its chemical composition. Bauxites which have a melting point of 1910 deg. C. are frequently used. It is possible by proper mixing to produce a bauxite brick which has a higher melting point than fireclay brick. It is necessary, first, to burn the bauxite at a high temperature and for a long period in order to obviate all shrinkages afterward. One part of the calcined

[1] *Records* Geological Survey of India, XXXII, 2, p. 175 *et seq.*
[2] *Tonindustrie Zeitung*, 1905, No. 91

bauxite is then ground fine and mixed thoroughly with the bonding fireclay, while the other part of the bauxite is ground like grog and used as such.

The Refractory Uses of Bauxite.

By A. J. Aubrey.

The crude bauxite is washed at the mines to remove some free silica. It is next calcined at a temperature of 2500 deg. F.; during this process it gives off its "chemical water," amounting to about 30 per cent. of the raw ore, and it also undergoes great shrinkage. It has been observed that the bauxite shrinks but little until after 2390 deg. F. is reached. Accordingly the lowest temperature limit at which bauxite should be calcined is 2500 deg. F.

An analysis of washed and calcined bauxite from Arkansas gave the following results: Mechanical water, 0.88 per cent.; silica, 6.40 per cent.; oxide of iron, 1.43 per cent.; alumina, 87.30 per cent.; titanium oxide, 3.99 per cent.

The calcined material can be bonded with fireclay, sodium silicate, or lime and made into brick or tile. As little as 4 per cent. plastic fireclay can be used for a bond for hand-made brick. When bonded with lime, the bricks become hard a few hours after making. This setting or hardening is probably due to the formation of a calcium silicate between the lime and the free silica, in analogy with the setting of silica brick when bonded with lime. After careful drying, the bricks are burned in down-draft kilns at a high temperature, when they become hard and tough, and can be subjected to rough treatment without breaking. A brick 9x2½x4½ in. weighs 7½ lb. and stands a crushing test of 10,000 lb. per sq. in.

Washed granular bauxite can not be successfully calcined in a vertical-shaft kiln, because it will pack and clog on account of its fineness, and so obstruct the passage of the kiln gases. On the other hand, it is not feasible to calcine it in the ordinary down-draft kiln on account of the perforated floor. I believe that to calcine it on a commercial scale would require a rotary kiln, using oil, gas, or powdered coal for fuel, preferably the first. Provided the cost of fuel is low at the mines, such kilns should be installed there in order to save the cost of transportation which is now paid on the 30 per cent. of water that the crude bauxite contains.

For basic open-hearth steel furnaces, a brick high in alumina and low in silica is required. A brick of this character is obtained in the following way: A pure white variety of the pisolitic bauxite (which has already been washed at the mines and has had a portion of its free silica removed) is selected and sieved. The fine material passing through the sieve is rejected, as it contains the greater part of the silica. The "piso-

lites," pebbles about the size of peas, are retained as they are higher in alumina than the rest of the bauxite. By using, as a bond, lime free from silica, a brick can be procured which will contain as low as 6 or 8 per cent. silica.

A high silica-content has always been the chief objectionable feature of bauxite brick for basic open-hearth steel furnaces, because the silica is attacked by the basic slag. Authorities on open-hearth practice have generally declared that bauxite brick would be suitable for basic open-hearth furnaces, provided a brick could be made with less than 12 per cent. silica. Recent tests bear out this statement.

A test was made several months ago in one of the basic open-hearth furnaces of the Bethlehem steel works in which a bauxite brick and a magnesite brick were placed side by side near the gas- and air-ports, and were submitted to the highest temperature attainable in the furnace. The magnesite brick bent and showed viscosity after seven minutes, against 15 minutes for the bauxite brick. Again, a magnesite brick and a bauxite brick were submerged in the slag near the doors for some time, after which they were withdrawn, and examined when cold. The magnesite brick was incorporated with slag, while the bauxite brick, when broken open, showed that the slag had not penetrated to its center, but had remained as a coating over the outside. Both bricks were badly affected, but the bauxite withstood the action of the corrosive slag as well as the magnesite. In more recent tests, bricks having a lower content of silica have shown up as well as the magnesite, when exposed to the action of the basic slag.

The comparison is hard to make in any case, because of the fact that either a bauxite or a magnesite brick, if left in the hearth of the furnace for any length of time and submerged in the slag, is so violently attacked that differences between the remnants taken out are not easily detected. Even a magnesite brick if thrown into a basic open-hearth furnace will be entirely eaten away after a considerable length of time, so that the only real test of bauxite will be to build a hearth (or portion of hearth) of that kind of brick, and test it, covering it with calcined bauxite and treating it in every way as the magnesite hearths are treated. Steps are now being taken to make tests of this character.

Sir William Siemens found that bauxite is a superior furnace lining, and though he used an inferior bauxite containing as much as 35 per cent. oxide of iron, he claimed that it lasts five or six times as long as the "Stourbridge first brick." Siemens also says of bauxite: "It is important to observe that bauxite, when exposed to intense heat, is converted into a solid mass of emery of such extreme hardness that it can hardly be touched by steel tools and is capable of resisting mechanical, as well as the calorific and chemical, action to which it is exposed."

Bischof, in his *Die Feuerfesten Thone,* says that bauxite, when not impure on account of the admixture of foreign substances, especially of iron, which generally occurs in considerable quantities in compounds of aluminum, is extremely refractory. He also goes on to say: "The addition of varieties free from iron, or the white ones, to other refractory clay offers the only important means known of increasing their percentage of alumina, and at the same time their refractoriness."

In addition to its use for open-hearth, bauxite brick has shown up successfully in other lines. Two recent applications concern such uses; first, as a lining for rotary portland cement kilns, and, second, as a lining for lead-refining furnaces.

As a lining for a rotary portland-cement kiln it has shown unusual durability and has given excellent service. In an experiment, the hot zone (about 10 or 12 ft.) of a 60-ft. rotary kiln, fired with coal dust, was lined with a 6-in. bauxite lining; the kiln had previously been lined with 9-in. fire-brick blocks.

A block for lining rotary portland-cement kilns in the hot zone must possess the following qualities: It must be neither too hard nor too soft. If too hard, it will not allow the cement coating to stick to its surface; if too soft, it will not hold the coating, but the latter will pull off, bringing portions of the brick with it. The cement coating affords an excellent protection for the lining, and if the bricks do not hold this coating they soon burn out. The bauxite block made for this purpose has all of the aforesaid qualities. After 10 months' continuous service, night and day, this 6-in. lining is still doing the work.

When one of these rotary kilns is shut down the loss of output in cement per 24 hours is equivalent to $250. As the minimum time required for lining or patching a kiln in the hot zone is from 36 to 48 hours, the loss of output will be between $375 and $500. It is apparent that a superior lining will be the cheaper, in the long run. Moreover, it must be remembered that only the hot zone need be lined with a bauxite block.

The second, and most recent, application of bauxite brick is that of lining portions of lead-refining furnaces. Pig lead (containing copper, antimony, occasionally arsenic, and other metals in small quantities) is charged into the hearth of the refining furnace, and melted by the reverberation of hot gases. The hearth of the furnace (10x13 ft.) is lined on the bottom by 9-in. square brick, set on end. The sides of the furnace are lined with a single course of 9-in. brick, laid with ends against the walls. During the process of refining, a scum rises to the surface of the molten lead; this scum contains most of the impurities which the refining is designed to remove. Inasmuch as the scum is composed of highly basic oxides, it was considered reasonable to substitute a basic lining for the fire-brick lining; accordingly, a lining of bauxite was put

in wherever the bricks were exposed to the slag. The result was that the bauxite brick lasted from five to six times as long as the ordinary fire-brick lining.

With these uses, and its possible advent into open-hearth practice and elsewhere, bauxite brick will prove a valuable refractory material.

BISMUTH.

THERE were no new developments of importance in the bismuth industry in 1905. The production of the metal continues to be controlled by the European combination, especially the firm of Johnson, Matthey & Co., of London, and the American consumption is chiefly through two or three pharmaceutical concerns in New York, Philadelphia and St. Louis. The consumption of bismuth for alloys is comparatively small. The total consumption of bismuth in the United States, as represented by the imports, is, however, considerable, as appears in the following table:

Year.	Pounds.	Value.	Av. per lb.
1896	124,263	$ 90,950	$ .73
1897	151,374	172,236	1.14
1898	137,205	162,846	1.19
1899	176,668	208,197	1.18
1900	180,433	246,597	1.37
1901	165,182	239,061	1.45
1902	190,837	213,704	1.12
1903	147,295	235,199	1.60
1904	185,905	339,058	1.82
1905	148,589	318,007	2.14

This shows the possibility of establishing in the United States a minor industry of considerable importance. The consumers have been deterred, however, partly by ignorance and partly by lack of confidence in the domestic supply of ore, while outsiders are fearful of antagonizing the ring, which in this industry is very strong. So long as this condition exists, therefore, anyone who has bismuth ore is dependent upon the ring for its market, and the latter is limited arbitrarily in view of the liberal supply offered in several countries. The various discoveries of ore which have been made in the United States have not, consequently, been developed. The Ballard mine at Leadville, Colo., has continued to produce bismuth-lead ore (about 12 per cent. Bi) from streaks which are occasionally found in that mine. Bismuth ore is also reported to have been found on Pine Creek, a few miles below Granite, Colo. A recent attempt in Arizona to extract bismuth by a wet process has not, so far as we are aware, amounted to much.

The total quantity of bismuth ore mined and marketed at Leadville, Colo., in 1905 was 9.2 tons, assaying from 5.4 to 13.9 per cent. bismuth.

It is not unlikely that a new source of bismuth will soon be developed in the United States through the electrolytic lead refinery which is now being erected by the DeLamar Copper Refining Company at Chicago, Ill. The Betts process will be used. Bismuth is the most objectionable

impurity in lead intended for corroding, but it is also one of the most difficult of removal by the Parkes process. Electrolytic refining, however, effects a clean separation, and it is possible to recover the bismuth from the anode slime, if it be present in sufficiently large quantity. The average base bullion is probably too low in bismuth to make attempt at recovery of the latter worth while, but it may be practical to smelt separately charges of ore high in bismuth and produce a base bullion comparatively rich in that metal, which may then be refined in special vats.

Bismuth ore is produced in Bolivia, and that country promises to be an important supply. There is even a prospect that metal from that source may be offered in competition with the trust.

The ordinary methods of bismuth smelting are described in THE MINERAL INDUSTRY. Vol. VIII, p. 58.

The price for metallic bismuth averaged high in 1905, as will be observed from the table on the previous page. In September there was a decline to 9s. per ℔. at London, and late in December there was a sharp break to 5s., the New York equivalents being \$2.25 and \$1.25 respectively. Bismuth salts declined accordingly, the price for subnitrate falling from \$2.45 per ℔. to \$2.25 in September, and to \$1.50 in December.

BORAX.

ALMOST the entire production of borax in the United States during 1905, as in previous years, came from San Bernardino, Ventura and Inyo counties, California, although surface deposits in Nevada near the California line now produce a small quantity. The production for a series of years is stated in the following table:

PRODUCTION OF BORAX IN CALIFORNIA. (*a*)
In tons of 2000 lb.

Year.	Tons.	Value.	Year.	Tons.	Value	Year.	Tons.	Value.
1894.....	5,770	$807,807	1898....	8,300	$1,153,000	1902....	(*b*)17,202	$2,234,994
1895.....	5,959	595,900	1899....	20,357	1,139,882	1903....	34,430	(*c*) 661,400
1896......	6.754	675,400	1900....	25,837	1,013,251	1904....	45,647	(*c*) 698,810
1897......	8,000	1,080,000	1901....	7,221	982,380	1905....		

(*a*) Reported by the California State Mining Bureau. (*b*) Mostly refined borax, whence the apparent discrepancy in value. Output of the other years is given as crude material. (*c*) Spot value.

Between two and four tons of crude material are consumed in making one ton of pure anhydrous boric acid. Of the material shipped from the mines, about six-sevenths contains not more than 25 per cent. borax; the other seventh is partly concentrated, but none of it is refined. For this reason, the amounts and the spot values of the crude product, beginning in the above table with 1903, offer the most suitable comparison.

The Borax Market.—No marked changes occurred during 1905 in the prices of either crude or refined borax; the refined material was a trifle lower than in the previous year. The New York market prices were for refined borax, about 6½@6¾c. per lb. delivered, which is about ½c. per lb. lower than in the previous year. Chicago pays less than New York, although the material has to be shipped from California to New Jersey, refined, and then shipped back to Chicago.

The refiners of borax in the United States are as follows: Borax Consolidated (Ltd.), Bayonne, N. J.; Pfizer & Co., Brooklyn, N. Y.; Brighton Chemical Company, New Brighton, Pa.; Thos. Thirkelson & Co., Chicago, Ill.; and Stauffer Chemical Company, San Francisco, Cal. A small quantity of borax is refined by producers in California and Nevada, but this product does not find so ready a market as that handled by the larger refineries.

California.—According to Charles G. Yale, in a preliminary report to the U. S. Geological Survey, the colemanite deposits in San Bernardino county are the largest producers in California, and the deposit owned by the Borax Consolidated, Ltd. (Pacific Coast Borax Company) is much the greatest of these. The American Borax Company, in San Bernardino county, is the second producer in importance. This company is working

a low-grade borate of lime, from which boric acid is produced by a special process described on a following page. The Western Mineral Company, which began operations only near the end of 1905, is working the same class of material. The Palm Borate Company also owns properties in the neighborhood of Daggett.

In Ventura county the Frazier Borate Company is mining material which is shipped to San Francisco and refined by the Stauffer Chemical Company. The Columbus Borax Company is also operating a deposit in the same county.

Near Big Pine, Inyo county, the Western Borax Company is working marsh dirt or mud that carries from 8 to 10 per cent. of borax. The output in Inyo county is limited by reason of the difficulties of transportation and recently by the scarcity of white labor. Activity in the gold mines of that district lured away most of the white men last year, and Chinese laborers had to be substituted.

Along the Mohave desert and the Death Valley region of California are numerous small deposits of low-grade borax, which will doubtless be utilized in the future. The "cotton ball" surface deposits near Winnemucca, Nevada, are being worked to a limited extent, the earth carrying about 15 per cent. of borax.

The most of the borax of this country comes from the colemanite mines of the Pacific Coast Borax Company, at Borate, San Bernardino county. The output here increased during 1905, and the roasters at Marion are now running at double capacity because more low-grade material than formerly is being mined. All the ore that averages 35 per cent. or more is shipped East directly, but the lower grades are first concentrated at Marion. Increased demand for borax warrants a greater output from the mines.

The company's improvements to its property will have a decided effect on the borax industry in the United States. It is building a railroad from Ludlow, San Bernardino county, on the Santa Fé main line, to Amargosa Valley in Inyo county, near Death Valley, by which to transport borax from the Lila C. mine. Extensive developments have been progressing for several years in this mine, where there is a body of colemanite of fine quality which will be shipped on completion of the railroad (131 miles) expected in July. It will open up a promising mining region before it reaches the borax property for which it is intended; it may be extended to Bullfrog, Nev., where it will handle the ores from other mines in that region.

The Pacific Coast Borax Company also owns borax properties in Nevada, including Teel's Marsh, which is now nearly worked out. Among other properties owned by the company are several in and around Death Valley, which have not yet been worked, but can be, if desired, on completion

of the new railroad. There is very little profit in manufacturing borax on a small scale from low-grade material.

Borax Consolidated, Ltd.—The seventh annual report of this London corporation, one of whose subsidiaries is the Pacific Coast Borax Company, covered the year ending Sept. 30, 1905. It showed a profit during that period of £254,025, a slight increase over the preceding year. For depreciation of plant and buildings, £15,000 was set aside, and £10,000 was written off for the closing of Connah's Quay Works and transference of its processes to another part of the plant. A further allowance of £20,000 was written off for exploration, inspection and development. A dividend of £1 per share was declared, or at the rate of 17½ per cent. for the year.

Progress in the Technology of Borax.

John Winkler, in the *California Journal of Technology*, states that the first borax made in the United States was marketed in 1864 in Lake county, Cal., where 12 tons were made by evaporation of the water from the lake. The price was $780 a ton. Borax now sells for $140 a ton, the annual United States production being valued at about $2,500,000. It is marketed as crude, refined borax and boric acid.

About 600 or 800 tons of borax are produced annually in the dried-up marshes of Nevada and Oregon, where it occurs as natural borax and simply requires leaching, evaporation and re-crystallization.

Boric acid is made principally from calcium borate and is used as a preservative and in medicine, in the manufacture of borax, and as a flux in glazing and enameling works.

The process of the American Borax Company, near Daggett, Cal., depends upon the fact that the affinity of boric acid for lime is not very great; acid fumes, such as SO_2, decompose borate of lime readily and completely, with the formation of salts of lime and free boric acid. Free SO_2 gas is forced into the aqueous pulp until all the boric acid is liberated. The acid is then leached out and the solutions run to large solar vats for evaporation. There are frames in the solar vats with horizontal shelves. The liquor to be evaporated is pumped on top of these by windmills and allowed to flow slowly down over the shelves. When sufficiently concentrated (about 4 or 5 oz. to the gallon), it is sent to crystallization vats. The crystals are dried on platforms to about 90 per cent. purity and are then ready for shipment.

By this process the American Borax Company made, last season, some 600 tons of crystallized boric acid from an ore containing about 10 per cent. Among the advantages of the SO_2 process are that SO_2 is selective, first attacking the borates in the ore; it saves the expense of a sulphuric acid plant; borax is much more easily and cheaply made from boric acid than from borate of lime.

Determination of Boric Acid.—The same author gives the following analytical methods for the determination of boric acid:

First Method.—Just enough of concentrated H_2SO_4 is added to a weighed quantity of ore in a flask to make a paste. This is then heated to drive out CO_2, and then distilled in the steam of the water bath or steam jet (not flame) with at least six 10 c.c. portions of wood alcohol. The receiver should contain a volume of pure glycerin equal to one-third of the volume after titration. The glycerin should contain 5 to 8 per cent. of water in order to make phenolpthalein more effective. The receiver is then titrated with phenolphtalein and about N/15 NaOH. A blank (with the ore left out) should be run through to obtain the amount to be subtracted for coloration, and a standardization in which the ore is replaced by a weighed amount of pure boric acid. A sharp end point is obtained. The residue in the distillation flask should be small, and should be burned with more alcohol to see if all the boric acid has been carried over. Repeat if necessary with a greater number of 10 c.c. portions of wood alcohol. This method cannot be used wherever the concentrated H_2SO_4 liberates acid fumes other than CO_2; i. e., it cannot be used with chlorides, nitrates, sulphites and similar salts.

Second Method.—Where the above method is inapplicable, and for liquors, the weighed ore or a measured volume of liquor is decomposed by concentrated H_2SO_4, made up to a definite volume and mixed uniformly by agitation. Almost neutralize the filtrate by solid carbonate of soda, boil 3 or 4 min. with an excess of solid $BaCO_3$, filter, take an aliquot part made slightly acid with H_2SO_4, boil again 2 or 3 min., cool and make exactly neutral to methyl orange. Glycerin is now added and the solution titrated with N_5NaOH, using phenolpthalein as indicator.

Third Method.—The iodate method of Jones, given in full in Sutton, is highly recommended by at least two efficient analysts.

BROMINE.

The production of bromine in the United States in 1905 amounted to 899,434 lb., exceeding the remarkable output of the previous year, but not by the same increment as 1904 surpassed its predecessor. Of the output in 1905, a little more than half was contained in bromides of the alkalies, in which form about four-fifths of Michigan's product is recovered. The output of the other districts is obtained as free bromine. In the following table, the yield of bromides has been calculated to its equivalent amount of bromine:

PRODUCTION OF BROMINE IN THE UNITED STATES.

(In pounds.)

Year	Michigan (a)	Ohio and Penna.	West Virginia	Total	Metric Tons	Value	
						Total	Per lb.
1896	42,000	367,450	149,835	559,285	254	$143,074	26c
1897	147,256	241,939	97,954	487,149	221	136,402	28.
1898	141,232	226,858	118,888	486,978	220	136,354	28
1899	138,272	193,518	101,213	433,003	196	125,571	29
1900	210,400	196,774	114,270	521,444	237	140,790	27
1901	217,995	227,062	106,986	552,043	250	154,572	28
1902	226,452	194,086	93,375	513,913	233	128,742	25
1903	320,000	180,000	97,000	597,000	271	170,145	28½
1904	646,249	147,807	85,256	879,312	399	215,431	24½
1905	579,434	223,000	97,000	899,434	408	139,492	15½

(*a*) Includes the bromine equivalent of the bromides recovered.

It is noteworthy that Michigan, which produces all the bromides that are made in this country from the original brine, showed a decrease, while the other districts, which make only bromine, recorded an increased output.

The producers of bromine in the United States are as follows: The Dow Chemical Company, Midland, Mich.; St. Louis Chemical Company, St. Louis, Mich.; Saginaw Salt Company, St. Charles, Mich.; John A. Beck Salt Company, Allegheny, Pa.; J. L. Dickinson & Co., Malden, W. Va.; Hope Salt and Coal Company, Mason, W. Va.; Liverpool Salt and Coal Company, Hartford, W. Va.; Hartford City Salt Company, Hartford, W. Va.; Syracuse Salt Works, Syracuse, O.; Coal Ridge Salt Company, Pomeroy, O.; Buckeye Salt Company, Pomeroy, O.; Excelsior Salt Works, Pomeroy, O.; Slagel Salt Company, Pomeroy, O.; the Pomeroy Salt Association Company, Pomeroy, O.

Market.—The year 1905 was marked by depressed prices owing to German competition. In January, bromine sold at 16c. per pound,

and for the remainder of the year at 14 @ 15c. per pound. About 75,000 ℔. was sold, on contracts dating back to 1902, at 23 @ 28c. per pound, but all the rest brought the above-quoted low prices. The highest price paid for potassium bromide at New York, on wholesale lots of 50 ℔., was 17c. per pound f. o. b., but the average for the year was 15c.

CARBORUNDUM.

The production of carborundum in 1905 was 5,596,280 lb., against 7,060,380 lb. in 1904, a decrease of 20 per cent. The entire industry is in the hands of The Carborundum Company of Niagara Falls.

The growth of the demand for carborundum is shown by the following table:

PRODUCTION OF CARBORUNDUM IN THE UNITED STATES.

Year	Pounds	Metric Tons	Value
1896	1,191,000	539	$365,612
1897	1,242,929	564	153,812
1898	1,594,152	723	151,444
1899	1,741,245	790	156,712
1900	2,401,000	1,089	216,090
1901	3,838,175	1,741	345,435
1902	3,741,500	1,697	374,150
1903	4,760,000	2,159	476,000
1904	7,060,380	3,202	706,038
1905	5,596,280	2,540	559,628

Progress in the Technology of Carborundum.

The Carborundum Furnace.[1]—When a mixture of silica and carbon is heated to a sufficiently high temperature, silicon is reduced, and if a proper amount of carbon is present, silicon carbide is formed, according to the equation:

$$SiO_2 + 3C = SiC + {}_2CO.$$

At relatively low temperatures the silica-carbon mixture yields a greenish amorphous substance, but as the temperature is increased this is converted into crystalline silicon carbide, or carborundum. If the temperature is increased considerably beyond that of its formation, carborundum is decomposed, the silicon being expelled as vapor and the carbon left behind in the form of graphite. This decomposition takes place without fusion of the carborundum, and the graphite is left as a pseudomorph of the carborundum crystals.

Since carborundum is infusible, the furnace cannot be tapped as in the case of calcium carbide, and it is, therefore, difficult to devise any method by which the manufacture of carborundum can be made a continuous process. Then, since it is easily decomposed at high temperatures, some means of regulating the temperature of the furnace must be provided, so that it may be kept between the limiting temperatures of the formation and decomposition of silicon carbide. This condition indicates that an arc furnace would be unsuitable, for while it is undoubtedly possible to produce carborundum in a furnace heated by an arc, it would not be a practical or efficient method of manufacture. No doubt an arc may be used in the

[1] Abstracted from an article with this title by F. A. J. Fitzgerald, *Electrochemical and Metallurgical Industry*, Feb., 1906.

production of substances that are vaporized or decomposed at the temperature of the arc itself if, as fast as these are formed, they can be drawn off from the high-temperature zone.

In the earliest form of carborundum furnace an arc was used, the furnace charge surrounding two carbons, between which an arc was drawn. This was soon abandoned for a furnace in which the charge is heated by means of a suitable resister.

A carborundum furnace using a 750-kw. current is a rectangular fire-brick structure 23x10 ft. over all, on the ground, and 10 ft. high. Only the end walls, 16 ft. apart, are permanent, the side walls being torn down and rebuilt after every run. Through the center of each end wall passes a carbon terminal. The terminal is composed of 25 carbon rods, each 34 in. long and of 4 in. square section. These are grouped in five rows of five rods each, with a copper plate between every two rows, for connecting to the transmission cables. The side walls being laid up, the furnace is charged about half full of the sand-coke-sawdust mixture, leveled off, and a semi-cylindrical trench of 20 in. diameter is formed longitudinally to connect the terminals; the trench is then filled and heaped with granular coke, to form the resister, the ends of the resister, in contact with the terminals, being formed of finely powdered coke. The furnace is then heaped with charge, and the current turned on.

The temperatures of formation and of dissociation of carborundum being fixed, and the amount of heat emitted from the resister being proportional to its radiating surface, it follows that the diameter of the resister must be adjusted to the current employed. Then, as the cylindrical crust of carborundum accumulates, less heat is transmitted to the enveloping raw materials, and the central temperature rises to the dissociation point of carborundum. For this reason, after a cylinder of a certain size is formed, it is no longer economical to run the furnace. At all times it is necessary to preserve a thick layer of unchanged mixture outside the carborundum cylinder, since it forms an efficient heat insulator.

When the current has been cut off, this unchanged mixture is removed, until a greenish amorphous substance is exposed. Muhlhaeuser, after he had treated it with hydrochloric acid, caustic soda, water, hot oxygen and hydrofluoric acid, found it to have the composition:[1]

C	27.93
Si	65.42
Fe_2O_3 and Al_2O_3	5.09
CaO	0.33
MgO	0.21
	98.98

[1] *Journal* of the Franklin Institute, Sept., 1893.

This substance lies immediately outside the carborundum cylinder, and is only a few centimeters thick. The carborundum cylinder, on the other hand, is about 35 cm. thick, which indicates that the temperatures of its formation and decomposition lie much farther apart than do the formation temperatures of carborundum and of the amorphous material. In considering this, however, the relative heat conductivities of the two substances would also have to be taken into account.

When the crystalline cylinder is removed from the furnace the granular coke, forming the resister, is found to be graphitized. The graphitized coke may be used as the core of another furnace, and will have a much lower resistance than a resister made of ordinary coke. When a furnace is built with a resister of ordinary coke it is necessary to use a high voltage at the start, and even then it may be some hours before the furnace comes to load. Then follows a great reduction in the resistance, due not merely to the heating of the carbon, but also to its conversion into graphite.

In attempting to compute the efficiency of the carborundum furnace, Prof. J. W. Richards[1] assumed a temperature of 3000 deg. C. and certain values for the specific heats of the substances; he then concluded that the heat necessary for raising the charge of the furnace to the required temperature is 42 per cent. of the heat energy generated, and that the chemical reaction takes 34.5 per cent., so that the efficiency is 76.5 per cent. Unfortunately, he failed to take into account the amorphous material that is also formed in the furnace, up to 14 per cent. of the carborundum. In actual practice, also, it happens almost invariably that a considerable quantity of the carborundum formed in the furnace is decomposed. Finally, the mixture always contains a large quantity of saw-dust, and usually a considerable quantity of water, of which no account was taken.

In the present state of our knowledge of the working of electric furnaces it is not possible to make even approximate estimates of absolute efficiency, and we must, therefore, be content to compare production and energy. Thus, Mr. Acheson has stated that he was using 9 kilowatt-hours to produce 1 lb. of carborundum. According to Prof. Richards, The Carborundum Company in 1902 produced 7000 lb. of carborundum with an expenditure of 36,000 h. p. hours, that is, 3.8 kilowatt-hours per pound.

Carborundum as a Refractory.—The use of carborundum as a refractory is one of the more recent developments, although from the very first its heat resistance was recognized as a quality equal in value to its hardness. As yet, its principal utilization has been to form a refractory coating on the firebrick lining of furnaces.[2] For this purpose the carborundum is finely ground and mixed in the proportion of three parts by weight of carborundum to one part of sodium silicate (water-glass). After thoroughly

[1] *Transactions* of the American Electrochemical Society, Vol. II, p. 57.
[2] *Electrochemical and Metallurgical Industry*, Vol. III, No. 4.

brushing the newly laid firebrick (the mixture will not stick to a surface which has been already fired), the carborundum is painted on to the depth of ½ mm. It is then left for 24 hours to dry, and afterwards the firing should be gradual. The fine particles of carborundum thus become cemented over the whole surface of the lining, cracks and all, and if properly done, it adheres firmly. Carborundum from the core of the electric furnace is infusible at 3900 deg. C., and is unaffected by oxygen, ozone, or sulphur at 1600 deg. C. It is also unaffected by chlorine, bromine, or boiling concentrated sulphuric acid, gaseous hydrochloric, nitric or hydrofluoric acid.

A welding furnace at Dusseldorf, treated in this way, worked daily from 9 to 7 for nearly six months without showing any deterioration in the lining.

Where basic slags or basic materials are encountered, fire-clay is employed as a binder, instead of the water-glass, the proportion being usually six parts by weight of carborundum to one part of fire-clay.

Carborundum, mixed with fire-clay in equal parts, has been used for repairing cracked gas retorts with considerable success. This was done while they were still hot, and they remained gas-tight for a long time.

For pointing the joints in brickwork of by-product coke ovens, a mixture of carborundum and fire-clay has been found most useful. The mixture is tamped in to about ½ in. deep, and completely closes the oven chamber against the flues, so that there is no entering of gases through the heating walls.

Carborundum has also been used for lining iron and other metals. In this case, three parts by weight of carborundum are mixed with two parts of sodium silicate. Where it is used for this purpose, however, the metal must remain constantly heated, as otherwise the coating will burst off when the metal cools.

To form shaped blocks of carborundum, Siemens & Co., of Germany[1], mix carborundum with free silicon, mold it, and then treat it in an electric furnace. The blocks thus made have all the resisting properties of carborundum.

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[1]British Patent, No. 21,347 of 1905.

CEMENT.

By Robert W. Lesley.

The cement industry of the United States recorded a pronounced advance in production during 1905, as well as a vigorous confirmation of the tendency first noticeable five years ago, to substitute portland for natural cement. While the output of portland cement in 1905 surpassed that in 1904 by 36 per cent., the amount of natural cement made in 1905 was less by 8.5 per cent. than in the preceding year. Not only did the output of portland in 1905 exceed its predecessor by the largest increment ever recorded, but it surmounts a long ascending series of annual productions, while the output of natural cement shows a steady decline from year to year. The total amount of cement of all kinds made in 1905 exceeded that in 1904 by over 29 per cent., as against an increase of 6 per cent. recorded by 1904 over its predecessors.

PRODUCTION OF CEMENT IN THE UNITED STATES. (*a*)
(In barrels.)

Year	Portland.			Natural Hydraulic.			Slag Cement.			Total.	
	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.
1896	1,577,283	$2,502,479	$1.59	7,407,311	$4,385,962	$0.59	*Nil.*	$........	......	8,984,594	$ 6,888,441
1897	2,430,903	3,724,905	1.54	7,890,573	3,976,050	0.50	40,000	60,000	1.50	10,361,476	7,760,955
1898	3,584,586	6,168,106	1.72	8,161,078	3,819,995	0.47	157,662	235,721	1.50	11,903,326	10,223,822
1899	5,805,620	10,441,431	1.80	9,686,447	5,058,500	0.52	244,757	360,800	1.47	15,736,824	15,786,789
1900	8,482,020	9,280,525	1.09	8,383,519	3,728,848	0.45	446,609	567,193	1.27	17,312,148	13,576,566
1901	12,711,225	12,532,360	0.98	7,084,823	3,056,278	0.43	272,689	198,151	0.73	20,068,737	15,860,731
1902	17,230,644	20,864,078	1.21	8,044,305	4,076,630	0.50	478,555	425,672	0.81	25,753,504	25,366,380
1903	22,342,973	27,713,319	1.19	7,030,271	3,675,520	0.50	525,896	542,502	1.03	29,899,140	31,931,341
1904	26,505,881	23,355,119	0.90	4,866,331	2,450,150	0.50	303,045	226,651	0.75	31,675,257	26,031,920
1905	36,038,812	33,326,523	0.92	4,473,049	2,413,052	0.54	382,447	272,614	0.71	40,894,308	36,012,189

(*a*) Statistics of 1900 and subsequent years, except 1905, are from the "Mineral Resources of the United States."

STATISTICS OF CEMENT IN THE UNITED STATES.
(In barrels of 380 lb.)

Year.	Production.		Imports.		Exports. (*b*)		Consumption.	
	Barrels.	Value.	Barrels.	Value.	Barrels.	Value.	Barrels.	Value.
1896.........	8,984,594	$ 6,888,441	2,809,172	$3,394,426	67,805	$ 103,315	11,725,961	$10,179,552
1897.........	10,361,476	7,760,955	2,200,871	2,688,122	62,761	103,389	12,499,586	10,345,688
1898.........	11,903,326	10,223,822	2,119,880	2,624,228	55,969	98,121	13,967,237	12,749,929
1899.........	15,736,824	15,860,731	2,219,246	2,858,286	116,079	213,457	17,839,991	18,505,560
1900.........	17,312,148	13,576,566	2,512,300	3,330,445	147,305	289,186	19,677,143	16,617,825
1901.........	20,068,737	15,786,789	994,624	1,305,692	303,380	752,057	20,759,981	16,340,424
1902.........	25,735,504	25,366,380	2,100,513	2,582,281	367,521	575,268	27,486,496	27,373,393
1903.........	29,899,140	31,931,341	2,439,948	3,027,111	312,163	466,140	32,026,925	34,492,312
1904.........	31,675,257	26,031,020	1,101,361	1,383,044	816,640	1,158,572	31,959,978	26,256,392
1905.........	40,894,308	36,012,189	891,134	1,102,041	1,060,054	1,428,489	40,725,388	35,685,741

(*b*) Includes re-exports of foreign cement.

The Cement Industry in the United States.

In reviewing the cement industry for the year 1905, the first fact that appeals to the mind is the enormous growth of the production of portland cement in the United States. The production in 1904, which was not con-

sidered in any sense a specially good year for building, attained the enormous volume of 26,505,881 barrels, a gain of nearly 20 per cent. over the production for the year 1903. This percentage of increase was lower in 1904 than for several preceding years, yet the increase was no less than 4,162,908 barrels.

The following statement, issued by the United States Geological Survey, shows the production of hydraulic cement in the United States for the calendar year 1905. This statement is preliminary to the annual report on the production of cement, which is now being prepared in that bureau.

The total production of all kinds of hydraulic cement in 1905, including portland, natural rock, and slag or Puzzolan cements, was 40,984,308 barrels, valued at $36,012,189.

The improvement in prices was noticeable, and continued throughout the year. The total production and values for the preceding year were: 31,675,257 barrels, valued at $26,031,920. Comparison of totals shows an increase in 1905 of 9,309,051 barrels in production and $9,980,269 in value.

Of the above total amount of cement manufactured in the United States in 1905, 36,038,812 barrels were portland cement, with a value of $33,326,523; 4,473,049 barrels were natural-rock cement, valued at $2,413,052; and 382,447 barrels were slag or Puzzolan cement, valued at $272,604.

Another event recorded in 1905 is that, for the first time in the history of the American portland cement industry, has the export of American portland cement been greater than the imports of foreign portland cement. The Government figures for the fiscal year ending June 30, 1905, show imports of 996,718 barrels, valued at $1,276,597, as against exports of American portland cement of 1,067,284 valued at $1,484,795. This in itself is a remarkable tribute to the quality of the domestic product and to the enterprise of the American manufacturer.

In connection with the statistics of the industry, it may be noted that there has been of late years a falling off in the production of natural cement, the output of this material having in 1902 about struck its maximum figure —falling off about 1,000,000 barrels in 1903, and over 2,000,000 barrels in 1904. Thus, while the total output of hydraulic cement of all kinds in the United States increased only 1,776,117 barrels over 1903, the great gain was made in portland cement, which, as above shown, increased over 4,000,000 barrels. This condition, in view of the low prices that prevailed in the early part of 1905, is likely to be witnessed again in the relative outputs of natural and portland cements during 1905.

Distribution of the Industry.—For a number of years the Lehigh district —embracing the counties of Lehigh and Northampton, Penn., and Warren county, New Jersey—manufactured from 60 up to as high as 72 per cent. of the total amount of portland cement produced. In 1900, the 15 works in

that district manufactured 72.6 per cent. of the entire output. In 1904, the 18 works in this same district made about 54 per cent. of the country's production.

While a map of the United States in the early days of the cement industry would have shown very few portland cement works outside of the States of Pennsylvania and New York, and a map of five years ago would show but little capacity outside of the above States, and Illinois and Michigan, yet a map of the producing territory in 1905 shows works in almost every State of the Union. This fact is rather singularly exemplified in the West, where rates are high, railroads but few, and the cost of cement at point of consumption involves a freight charge of sometimes three or four times the price of the cement itself at the mill.

California a few years ago was one of the great importing States. Vessels from Germany and England, bound for the Pacific coast for grain cargoes to Europe, usually carried portland cement as ballast, or at extremely low rates of freight, and the western coast of the United States remained one of the largest markets for the European brands of cement. Today California has three works and one of these is in process of enlargement, so that it will become one of the great producers of the country and very likely will supply large quantities of cement to Mexico and Panama by reason of its easy access to the sea. With the development of mining and the installation of irrigation plants in Montana, Nevada, Wyoming, Utah and Colorado cement plants have been brought into existence in the latter two States, and in both cases developments are under way for largely increasing their capacity.

Another field originally exploited by the Iola Portland Cement Company, namely, the gas belt of Kansas, is today the center of an active cement industry, three works being already in existence and two more under construction, and with the cheap fuel and excellent limestone the farming section of the West, in North and South Dakota, Nebraska and Oklahoma, are being supplied with portland cement which is rapidly supplanting the present high-priced lumber. The Dakotas find cement works at Yankton, S. D., while Missouri is the center of a large industry along the Missouri river at Hannibal and St. Louis. Of the older developments it is hardly necessary to speak, beyond stating that Michigan has grown into a large producer, while both Indiana and Illinois are today centers of rapidly developing portland cement production.

The old Louisville district, which produced so many millions of barrels of natural cement, has at last come to realize that portland cement is the product of the future, and, under the auspices of those connected with the natural cement industry, a large works is now under construction in Kentucky, while at Kosmosdale, near by, another plant is in operation, controlled by Eastern capital. The Southern States show some signs of

development, works in Georgia, Alabama, Texas and Virginia being already in operation and other works, at the seaboard at Norfolk and at Nashville, Tenn., being projected.

Materials.—As the first successful manufacture of portland cement in this country was confined for some time to the Lehigh district, it is but natural that raw materials of the character found there should have been sought by those engaged in the business in other parts of the country, and, where soft limestones with soft shales or cement rocks have been found, excellent results have been achieved. The development of productive works in the marl and clay districts has not been so rapid, but the investigators in this field are discovering new processes and new methods for increasing kiln output with low fuel consumption.

Machinery.—In European works, and also in some of the Western works in this country, the use of longer kilns has been marked for many years, and the practice has been followed in other parts of this country, the tendency of leading manufacturers being to increase the length of the kiln with a slight increase in the diameter. The results, in a general way, may be said to produce an increased output per kiln, with a very slightly diminished coal consumption. In some of the new works, kilns running from 100 to 150 ft. long are installed. In crushing and grinding machinery, modern practice seems to tend to a more gradual reduction of the rock, as well as of the clinker, by means of intermediate rolls and crushers. Air separators are also coming in more generally. Many improvements in leading forms of grinding machinery are noted for the year, among them the new three-roll Griffin mill, the new form of the Kent mill and also the improved apparatus in tube-mills.

The year 1905 seems to have marked the final disappearance of the non-fireproof mill, all modern construction of today being absolutely fireproof, made either of steel or concrete. This applies not only to mill buildings, but to stock-houses as well, so that from a very hazardous fire risk in the days of the old cement mills, the mill of 1905 has become practically a fireproof building, on which little or no insurance need be carried.

Growth of Standards.—An interesting feature of the past year has been the final adoption and issue to the public of the new specification for portland cement, formulated by the joint committee, of the American Society of Civil Engineers, the American Society for Testing Materials, the American Railway Engineering and Maintenance of Way Association, and the American Association of Portland Cement Manufacturers. This specification, which represents the work of the committee for several years, was finally established at the end of 1904, and so universal has been the appreciation of the standards recommended that no less than 20,000 copies of the specification have been sent out to consumers, engineers and producers in all parts of the country. On the lines of the joint committee

above mentioned is also the joint committee on concrete and reinforced concrete, which is charged with the preparation of standards in this new art. This committee is composed of representatives of the same societies, above named, who also had meetings with representatives of the National Board of Fire Underwriters; the National Fire Protection Association; the Concrete Block Machine Manufacturers' Association, and the National Association of Cement Users.

The work is no insignificant one, and the Government laboratory at St. Louis, in charge of the United States Government advisory board, is co-operating in making the experiments required. The work is likely to take several years, but its results will be most important, not only to the producers of cement, but to the great body of engineers, contractors and consumers who are interested or engaged in the art of making concrete and reinforced concrete.

Other associations on similar lines formed during the year are the Association of Cement Users, which held its first meeting in Indianapolis last year, where 600 users of cement in its various forms were present, and the Northwestern Cement Products Association, which also held a meeting early in the year, and was largely attended by those interested in cement in the Northwest. These bodies are all in close touch with the Association of American Portland Cement Manufacturers, whose scientific papers, produced during the past year, have been read and quoted largely all over this country and in Europe.

The Market.—The features of the market during 1905 were the extremely low prices prevalent during the early months. The early advent of winter in 1904, with the snow storms of November and December, had a tendency to close outdoor building construction very abruptly, leaving considerable stocks of cement on hand at the end of the year, so that manufacturers found themselves hampered with large quantities of cement with the winter months, continuous snow storms and a cessation of building operations confronting them. In view of the limited storage in the cement districts of the United States, the effect of this accumulation of material was to break prices and to cause some of the producers to market their material at almost any figure. A disease of this kind is contagious, and it has been repeatedly demonstrated that a floating surplus of a few hundred thousand barrels of cement in the spring will govern the market for the entire year. This certainly was almost the case for 1905, as the early spring contracts governing the bulk of the business for the year were made at a time when conditions were most inauspicious, when the demand for cement was practically nothing, and when stocks were accumulating at the mills owing to the unprecedented adverse weather conditions. In April, building operations were resumed on a large scale all over the country, though, owing to the closing of many contracts at low prices, the cement manufacturers

reaped but little benefit from this increased demand. This continued to be the case until June, when prices stiffened in many parts of the country, although in July there was again a falling off, which lasted for a short time, but in September consumption overtook production. In the latter months of the year the enormous demand, coupled with a car shortage all over the country, produced in some sections what was almost a cement famine.

The first effects of this cement famine, so to speak, were felt on the Pacific coast, where the local newspapers were filled with articles describing work held up because of the impossibility of securing cement, and where the arrivals of cars of cement from interior districts, or vessels with cement from abroad, were almost occasions for rejoicing. This condition of affairs spread from the West to the East, so that the shortage which began in California was again repeated in Utah and Colorado, which in turn were forced to buy large quantities of cement from works east of them in Missouri and Kansas.

The advance in the price of lumber, owing to the denudation of the forests, had—together with the great growth of wealth in the West—produced an enormous demand for portland cement for purposes of construction hitherto unthought of. As a natural consequence, the Western works were soon taxed to their utmost capacity and every mill was behind with its orders.

Eastern mills, which, during the early season, had been largely limited to the Eastern market owing to the cost of freight from the Lehigh region to the Western States, soon found a demand from all parts of the country for their product, and they, too, were promptly overloaded with orders. This condition prevailed all over the country during October and November and marked the close of 1905, which terminated with the lowest stock on hand at any similar period. It is to be regretted that an industry so important as that of portland cement should, by reason of the bad weather in the early part of its business year, have suffered such large losses as it has during 1905, caused by the extremely low prices made on account of the temporary necessities of the mills. Portland cement is as important in construction as lumber, steel and brick, is a staple commodity and has a steady and increasing demand. Manufacturers are now providing sufficient capital and sufficient storage capacity to take care of their outputs during the winter months, so that in dull seasons they may be able to hold their cement without sacrificing prices, not only on current sales, but on the enormous future requirements of the contractor for his work during the year.

The Cement Industry of Canada.

The Canadian cement industry in 1905 reflected strongly the same tendency towards the advance of portland at the expense of natural cement.

The production of natural rock cement which in 1904 had decreased to 56,814 bbl., valued at $50,247, fell off in 1905 to the comparatively small amount of 14,184 bbl., valued at $10,274. This was made by three firms in Ontario.

The production of portland cement, however, continues to increase steadily. Thirteen companies were operating plants during 1905 with a total daily capacity of about 8,000 bbl., viz.: one in Nova Scotia, two in Quebec, nine in Ontario and one in British Columbia, while another in Ontario was engaged in reconstruction.

The average price per barrel of portland cement in 1905 was $1.42, or a fraction of a cent higher than in 1904.

STATISTICS OF CEMENT IN CANADA.

Year.	Production (a).				Imports (b).			
	Natural Rock.		Portland.		Natural Rock.		Portland.	
	Bbl.	$	Bbl.	$	Bbl. (c)	$	Bbl. (c)	$
1900	125,428	$99,994	292,124	$ 562,916	3,742	$ 4,711	342,463	$ 498,607
1901	133,328	94,415	317,066	565,615	4,680	6,865	424,324	654,595
1902	127,931	98,932	594,594	1,028,618	7,786	17,755	518,846	833,675
1903	92,252	74,655	627,741	1,150,592	3,603	6,333	609,700	868,131
1904	56,814	50,247	900,358	1,272,992	3,181	5,391	651,681	995,017
1905	14,184	10,274	948,274	1,346,548	4,463	10,690	849,577	1,234,649

(a) Sales. (b) Fiscal year ending June 30. (c) Barrels of 380 lb. Exports of Canadian manufacture were valued at :$3,296 in 1900; $1,514 in 1901; $2,267 in 1902; $2,851 in 1903; $3,205 in 1904; $5,430 in 1905.

Of new enterprises, an important one is that of the High Portland Cement Company of Allentown, Penn., which has purchased 10,000 acres of limestone and clay lands in Thurlow township, near Belleville, Ont., and will at once erect a plant and begin manufacturing in the spring of 1906. In the western part of the Dominion, the Alberta Portland Cement Company is establishing at Calgary, Alberta, portland cement works to have an initial production capacity of 600 barrels per day, with provision for enlargement to an eventual capacity of 2000 barrels per day. The capital is being provided by those who are also chiefly interested in the Vancouver Portland Cement Company, operating cement works at Tod Inlet, near Victoria, Vancouver island, British Columbia.

All the portland plants in operation in Ontario, at the beginning of 1905, are described individually and in detail by P. Gillespie, in Part I of the Report of the Ontario Bureau of Mines for 1904, Vol. XIV, pp. 118-183; the same article contains technical information as to the manufacture, uses and testing of cement, and the preparation of various classes of concrete.

CHROMIUM AND CHROME ORE.

By Edward K. Judd.

The domestic production of chrome ore forms but an insignificant part of the consumption of that material in the United States. California is the only producing State, and its output is limited to the small requirements of a few smelters in the vicinity, which use the ore for furnace lining. Even if the large consumers in the East were unable to import their supplies cheaply, and free of duty, from New Caledonia, India, Turkey, and Canada, the domestic resources could not possibly supply any important proportion of the present demand for chrome ore.

STATISTICS OF CHROME ORE IN THE UNITED STATES.

(In tons of 2240 lb.)

Year.	Production. (a)			Imports.			Consumption.	
	Long Tons.	Value.	Value per Ton.	Long Tons.	Value.	Value per Ton.	Long Tons.	Value.
1896	786	$7,775	$ 9.89	8,869	$187,400	$21.13	9,655	$195,175
1897	*Nil.*	......	......	11,566	186,313	16.11	11,566	186,313
1898	*Nil.*	......	......	16,304	272,234	16.70	16,304	272,234
1899	*Nil.*	......	......	15,793	284,825	18.03	15,793	284,825
1900	140	1,400	10.00	17,542	305,001	17.39	17,682	306,401
1901	130	1,950	15.00	20,112	363,108	18.05	20,242	365,058
1902	315	4,725	15.00	39,570	582,597	14.73	39,885	587,322
1903	150	2,250	15.00	22,931	302,025	13.13	23,081	304,275
1904	123	1,845	15.00	24,227	348,527	14.38	24,350	350,372
1905	(b)150	(b)2,250	15.00	54,434	725,301	13.32	54,848	727,551

(a) Reported by the California State Mining Bureau. (b) Estimated.

Market and Trade.—The growing importance of New Caledonia as a producer of chrome ore continues to make itself felt. Prices have steadily fallen and reached their lowest level in 1905, viz., $17 per long ton, f. o. b. Atlantic ports, on the basis of 50 per cent. chromic oxide, with a premium of $0.40 per ton for each unit above that base. This is to be compared with $19 @ $19.75, the quotations of 1904. Freight rates from Noumea to New York are low, the ores being carried as ballast, and range from $2.88 to $4.32 per ton.

There have been no noteworthy advances in the commercial applications of chrome ore. The chief consumption is in the making of chromium salts used in tanning and dyeing works; the manufacture of chrome brick for refractory purposes (the crude material is also used in the same way); and the production of ferro-chromium alloys for use in special steels. The leading manufacturers of chromium salts are the Kalion Chemical Company, of Philadelphia, and the Baltimore Chrome Works, of Baltimore; the Harbison-Walker Refractories Company of Pittsburg controls the chromite brick-making industry.

Despite the rapidly growing importance of the consumption of chrome ore for the manufacture of ferro-chrome, chrome brick, and for use as refractory material, the chief consumption continues to be in the manufacture of chrome salts. This is a highly important branch of the American chemical industry. The technology of chromate manufacture was described in THE MINERAL INDUSTRY, Vol. IV, pp. 99-110. In the manufacture of 100 lb. of bichromate of potash there is required from 115 to 150 lb. of ore containing 50 per cent. Cr_2O_3. Perhaps the most important part of the process is the calcination of the ore to convert the chromic oxide (Cr_2O_3) into chromic acid (Cr_2O_3). This is done in reverberatory furnaces. As illustrating the magnitude of one of the American factories, it may be stated that the Baltimore Chrome Works has 12 furnaces, with hearths 10x60 ft., divided by steps into three sections, and grates varying from 33 in. x 7 ft. to 40 in. x 10 ft. and 36 in. x 10 ft., or say an average of 36 in. x 10 ft.=30 sq. ft. to 600 sq. ft. of hearth area, a ratio of 1:20. Each furnace has an independent chimney and burns 30,000 lb. of mine-run, bituminous coal in 6½ days, or 6.4 lb. per sq. ft. of grate per hour. Coal costs about $2.70 per ton.

Ferro-chrome.—The Willson Aluminum Company, of New York, with plants at Kanawha Falls, W. Va., and Holcomb Rock, Va., is the sole producer of ferro-chrome in the United States. The Kanawha Falls plant uses the energy from three 800-kw. generators, and that at Holcomb Rock uses one-third as much. At the former, the 110-volt, alternating current of 22,000 amperes is distributed among seven circular crucible furnaces. The combined capacity of the Willson Aluminum Company is 200 to 250 tons of ferro-chrome per month. The quality of the product is improving; at first it was soft and high in carbon, but now a dense alloy, containing not more than 5 or 6 per cent. carbon, is regularly produced. A certain consignment assayed as follows (No. I), two makes of French material (Nos. II and III) being given for comparison:

No.	Cr.	Fe.	C.	Si.	Al.	Mn.	Mg.	S.	P.
	%	%	%	%	%	%	%	%	%
I	72.70	21.40	5.30	0.60	....	.....	.....	.....	.....
II	65.30	27.25	6.10	0.85	....	0.415	.....	0.055	0.034
III	64.45	21.80	8.95	0.65	3.06	*Nil.*	0.43	*Nil.*	0.050

The production of ferro-chrome by the Willson Aluminum Company in 1905 was about 2000 short tons, as compared with 2500 tons in 1904. The decreased output was occasioned by the temporary stagnation of American war-ship building, awaiting the decision of the Congressional naval construction committee as to the design of the two new battleships recently authorized. During the latter months of 1905, and continuing into 1906, consumption of ferro-chrome for making armor plate ceased entirely, both the Bethlehem and the Carnegie works having accumulated sufficient

chrome steel to fill orders for some time. In the absence of home demand, the Willson company is exporting nearly its whole output to Sheffield, England, and to Japan. The use of chrome steel for other purposes than projectiles and armor plate is gradually increasing; rails, car-wheels and ore-crushing machinery are among the objects to which chrome steel can be applied.

The current price of ferro-chrome, carrying 70 per cent. chromium and not more than 6 per cent. carbon, is $185 per ton of 2000 lb. The percentage of chromium is the deciding element in the cost of production; owing to the affinity of chromium for carbon, to make a high-chromium alloy involves the rapid consumption of the electrode carbons, adding to the expense and at the same time tending to injure the quality of the product through excess of carbon.

Chromite in the United States.

California[1].—Up to the institution of the present tariff in 1897, the chromite production of California ranged between 1500 and 3500 tons per year; at present it is not more than 150 tons, and this is consumed in the immediate vicinity. Several counties have produced chromite in past years; the idle mines have not been exhausted, but await only a more active demand for their product. The following are the mines now active:

Calaveras county.—The Big Pine mine is owned by the Penn Chemical Works, which consumes the output for lining the reverberatory furnaces at its smelter at Campo Seco.

Fresno county.—The serpentine area six miles southwest of Toll House yields a quantity of low-grade ore which is mined by the Copper King, Ltd., for its own use.

Shasta county.—The mines on Shotgun creek, near Sims, which yield an ore high in chromium and low in iron, are still active, and are profitable by reason of their nearness to the railroad. The ore occurs in lenses in serpentine, from one of which 1500 tons has been taken. It is shipped as far as to Arizona, Colorado and Montana, and is also used for furnace bottoms at the Bully Hill and the Keswick copper smelters.

Sierra county.—D. E. Luse & Co., of Camptonville, Yuba county, are mining a deposit of magnesian chromite, which is used for furnace lining. The output is shipped by wagon to the railroad at Nevada City.

North Carolina[2].—Chromite is found in the peridotite areas of western North Carolina, but no localities have proved commercially valuable because of inadequate transportation facilities. New railroads, however, are threading the district, and will stimulate production.

A promising occurrence is at Mine Hill, Yancey county, five miles north of Burnsville. Numerous veins of chromite, up to 3 in. wide at the sur-

[1]Information obtained from the "Structural and Industrial Materials of California," published by the California State Mining Bureau, 1906.

[2]"The Mining Industry in North Carolina during 1904," Joseph Hyde Pratt.

face, have been uncovered; one of these veins widened to 3 ft. at the bottom of a 9-ft. pit. This deposit is owned by Garrett Ray, of Burnsville. At the time this region was prospected, Asheville, on the Southern Railway, 40 miles to the south, was the nearest railroad point; the recent extension of the Southern & Western from Erwin, Tenn., to Marion, N. C., now passes within four miles of the locality. A selected sample from Mine Hill showed the composition: Cr_2O_3, 58.00; Al_2O_3, 15.52; FeO, 14.45; MgO, 8.26; SiO_2, 3.20; CaO, 0.70 per cent.

In Jackson county, a peridotite area around Webster carries veins of chromite, which have been prospected to some extent. The nearest shipping point is Sylva, three miles north, on the Southern Railway. Near Balsam Gap, in the same county, veins of chromite have been uncovered close to the railroad. This property belongs to the Highland Forest Company, of Waynesville, N. C.

The United States Chrome and Nickel Company, organized in 1905 for the purpose of exploiting the alleged large deposits of chromium and nickel ores of Buncombe county, N. C., concluded that the quantity of ore available did not warrant any expense for development or erecting of concentrator. The enterprise was therefore abandoned.

THE WORLD'S PRODUCTION OF CHROME ORE. (*a*)
(In metric tons.)

	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Bosnia..........	443	396	458	200	100	505	270	147	279	(*c*)
Canada..........	2,125	2,393	1,834	1,824	2,119	1,156	,817	3,184	5,512	7,781
Greece............	1,600	563	1,367	4,386	5,600	4,580	11,680	8,478	15,430	(*c*)
New Caledonia (*b*)	16,018	9,054	14,300	12,480	10,474	17,451	10,281	21,437	42,197	51,374
Newfoundland....	1,031	3,084	657	717	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil*	*Nil.*
New South Wales.	3,914	3,433	2.145	5,327	3,338	2,523	454	1,982	403	53
Norway..........	*Nil*	*Nil.*	*Nil.*	41	165	85	22	*Nil.*	3	(*c*)
Russia............	6,682	13,433	15,467	19,146	18,233	22,169	19,655	(*c*)	(*c*)	(*c*)
United States....	799	*Nil.*	*Nil.*	*Nil.*	142	132	320	152	1 25	122

(*a*) From the official reports of the respective countries. (*b*) Exports. (*c*) Statistics not available at time of publication.

Of the exports of chrome ore from New Caledonia in 1905, the exports being equivalent to the production in this case, 46,000 tons were furnished by the Tiebaghi mines in the northwest of the island. These deposits, mostly alluvial, are easily worked, and are rich, the product averaging about 55 per cent. Cr_2O_3, without crushing, washing or other preliminary preparation. The spot price in 1905 was about £1 14s. ($8.16) per ton on basis of 50 per cent., and 2s. per unit in excess of that grade.

Comparatively little chrome ore is shipped from other parts of New Caledonia, but it is reported that they contain valuable deposits, which will be developed when those of the Tiebaghi mines become exhausted.

Metallurgy of Chromium.

Manufacture of Ferro-Chrome.[1]—Ores for the production of ferro-chrome should be as pure as can be had, and free from silica, since the latter has a

[1] O. J. Steinhart, "Notes on Metals and their Ferro-Alloys used in the Manufacture of Alloy Steels," Institution of Mining and Metallurgy, Jan. 18, 1906.

tendency to enter the alloy instead of going into the slag, when the ore is smelted in the electric furnace. A suitable ore contains not less than 50 per cent. chromic oxide. New Caledonian ores carrying 56 per cent. are readily obtainable at European ports for $16.80 per ton.

Of the following analyses of chrome ore, I is Turkish, and II is New Caledonian ore:

	Cr_2O_3	FeO Fe_2O_3	Al_2O_3	MgO	SiO_2	CaO	MnO	P_2O_5	H_2O
	%	%	%	%	%	%	%	%	%
I. Turkey	51.70	14.20	14.10	14.30	3.50	1.70	....	....	0.30
II. New Caledonia	55.70	16 60	16.20	9.80	0.20	0.25	0.20	0.05	1.05

At one time ferro-chromes, containing 50 to 60 per cent. of chromium, were made from chromite by reduction with charcoal in brasqued crucibles, adding suitable fluxes to slag off the gangue. Later it was made in small blast furnaces, but practically all is now made in the electric furnace.

When chrome steels first came into use the ferro-chrome employed for these steels was made at Krupp's Works, Essen, at Sheffield, in Sweden, at Unieux, France, and at Brooklyn, N. Y., in crucible furnaces capable of being raised to a high temperature. The alloys thus obtained were mostly low in chromium and high in carbon. In England by the Darwen & Mostyn Company, and also in France at the Forge de l'Adour, near Bayonne, chromite was run down in the cupola to a ferro-chrome of 60 to 70 per cent., often, however, containing as much as 10 to 12 per cent. carbon, and in this state it was sold for the high price of about $300 to $350 per ton.

Both the product and the method of manufacture were far from satisfactory, especially as, owing to the high fusing point of the alloy, the furnaces frequently froze up; therefore the cupola had to give way to the much more easily controlled electrical furnace. The change was stimulated when calcium carbide came to be produced in such large quantities that the supply exceeded the demand, and the works directed their attention to the application of the furnaces to the manufacture of steel and other metallurgical products.

A. Keller[1] stated that a ferro-chrome containing 60 to 70 per cent. Cr and 7 to 8 per cent. carbon is easily produced in a slightly modified calcium carbide (arc) furnace.

The cost of the manufacture of ferro-chrome (60 to 70 per cent.) is difficult to arrive at, and will naturally vary according to the cost of power, but taking the figure given by R. S. Hutton[2] at 1.21 h. p. per annum for 1 ton of the alloy, we arrive at the following:

2 tons of chromite (50%) at $16.80	$23.60
1.2-h. p. years at $19.20	23.04
Repairs and renewals, electrodes, labor, etc.	7.20
Total,	$63.84

[1]*Journal* of the Iron and Steel Institute, 1903, Vol. I, p. 162.
[2]*Journal* of the Society of Chemical Industry, 1905, p. 590.

Accordingly, ordinary ferro-chrome with about 6 to 8 per cent. of carbon can be made for, say, \$72 per ton, not including general expenses. It is selling at prices varying between \$110.50 and \$144 per ton on a 60 per cent. basis of chromium, according to carbon contents and purity.

Although, in the majority of cases, 6 per cent. and more of carbon in a ferro-chrome does not debar its use in many steels, still a number of steel makers will not use such an alloy when they desire to make low-carbon steel containing several per cent. of chromium. The Giffre Electro-Chemical Company has placed on the market some very pure alloys, as will be seen from the following two analyses:

	Cr. %	Fe. %	C. %	P. %	S.%	Si. %	O. etc., %
I.	62,450	36.702	0.555	0.031	0 009	0.115	0.141
II.	68,174	29.683	1.271	0.013	0.009	0.115	0.205

The value of alloy I is about \$720 per ton, and that of II about \$432 per ton, both on a basis of 60 per cent. with a corresponding increase for each per cent. above 60. The Giffre company claims that, apart from the low-carbon tenor of these alloys, they are more easily dissolved by the steel, whereas the high-carbon ferro-chromes cause segregation owing to the presence of carbide of chromium in these alloys.

Nothing can be learned as to the actual method of manufacture of these high-priced products, but already Moissan has shown that the following reaction takes place when treating chromium carbide with chromic oxide in the electrical furnace:

$$3\,Cr_3C + Cr_2O_3 = 11\,Cr + 3\,CO,$$

and further the writer has observed that the temperature and time of reduction have an important bearing upon the amount of carbon which is absorbed.

C. Coombs[1] describes Heroult's method for making low-carbon ferro chrome:

> The ordinary ferro-chromium ordinarily contains 9 to 10 per cent. of carbon, which is too large in quantity for certain uses; accordingly Heroult modified his furnace in the following manner: The metallic case of the crucible, which is coated with carbon, was replaced with ordinary refractory material or, better still, chromite. In place of a single electrode, two are used in series, so that the electric current passing down one electrode enters the crucible and then ascends the other carbon. By means of two voltmeters placed in shunt to each arc it is known if both arcs are established. With the new process the amount of carbon is reduced to between 2 and 6 per cent. It is possible to go further and reduce the carbon to 1 per cent. by adding an excess of chromite. The new type of ferro-chrome which is called "doux" is very different in appearance from that containing 9 to 10 per cent. C. It is white and compact, and with a marked crystalline fracture. Sometimes magnificent crystals are obtained.

In this case it can be seen that the excess of chromite plays the part of a decarburizer as above indicated. Recently E. F. Price has taken out a number of patents somewhat on the lines indicated, but no information as to practical results is yet obtainable.

The aluminothermic method of manufacture, applicable to quite a num-

[1] *Science Abstracts*, 558, pp. 244, 1905.

ber of other metals and alloys, has formed the subject of so many publications by Dr. H. Goldschmidt that it must be well known by now; it consists in mixing finally divided metallic aluminum with the oxide of the metal to be produced, in this case, Cr_2O_3, and igniting the mixture

$$Cr_2O_3 + 2\ Al = 2\ Cr + Al_2O_3.$$

The reaction, once started, proceeds rapidly and with the generation of great heat, resulting in a fused mass of metallic chromium of considerable purity. T. Doring[1] has found it to be composed as follows: Cr, 97.41; Fe, 1.38; Al, 0.16; Si, 0.73 per cent.

Another specially pure specimen, according to A. Mathews,[2] is as follows: Cr, 99.55; Fe, 0.14; Si, 0.31 per cent.

The price is somewhat high, being 54@60c. per lb., according to quantities, but Thermite, Ltd., claims certain advantages for this product, such as the uniform and easy way it alloys with steel, without giving rise to losses, which it states to be as much as 20 to 25 per cent. in the case of ferro-chromes. Reliable figures as to the world's production of ferro-chromes are difficult to arrive at, but should be somewhere between 4000 and 5000 tons per annum, the United States having produced as much as 2500 tons in one year.

Steel-making with Chromiferous Iron Ore.—H. H. Campbell, of Steelton, Pa.,[3] proposes to use iron ore containing from 1 to 5 per cent. chromium for the production of steel low in chromium. He has succeeded in producing a steel containing only 0.08 per cent. Cr. It is well known that, if steel has a content of chromium in excess of a certain small proportion, it is practicably unfit for use in engineering work; and also that it has not been practicable to make use of iron rich in chromium as a starting metal for the manufacture of steel on a profitable basis, especially on a large scale, because of inability to effect the economical removal of the chromium. Mr. Campbell's method is first to treat the chromium-iron in a basic bessemer converter, producing a basic slag, and then to oxidize the chromium by prolonging the blow beyond the usual period, which causes the chromium to enter the slag. The charge is then drawn from the converter into a ladle (having a device for drawing the metal from beneath the slag and then stopping or controlling the flow of slag). The subsequent treatment depends on the final use for which the steel is intended. For low-carbon steel, if the de-chromized metal does not contain much oxygen, it is incorporated with ferro-manganese to obtain the usual reaction; if the de-chromized metal contains considerable oxygen, it is charged into a second converter having a silicious lining, with an addition of unblown molten iron (free from chromium and having a higher carbon content), and is finally re-carburized.

[1] *Journal für Praktische Chemie*, 1902, p. 66-141.
[2] *Journal* of the Iron and Steel Institute, 1902, p. 185.
[3] United States patent No. 795,193.

Determination of Chromium in Steel.[1]—Volumetric methods have been devised by Ibbotson and Howden for the estimation of chromium in steel.

1. The sample is dissolved in a small amount of nitric acid of 1.20 sp. gr. and heated to expel nitrous fumes. After copious dilution, 2 to 3 grams of ammonium persulphate and about 0.01 gram of silver nitrate are added, and the solution heated until the chromium and manganese are completely oxidized. If the quantity of the latter metal be large, manganese dioxide may separate, and must be removed by filtration. The solution is cooled, heated with excess of ammonium acetate, and lead acetate solution is added. The precipitated lead chromate is collected on an asbestos filter, washed with dilute ammonium acetate solution, and then dissolved off the filter with nitric acid. After diluting the solution, an excess of standard ferrous sulphate solution is added, and the estimation completed by titration with N/20 potassium permanganate solution.

2. Steels containing large quantities of tungsten cannot be completely decomposed by nitric acid alone. In this case 0.5 gram of the sample is heated with 10 c.c. of sulphuric acid (1 to 4) until nearly dissolved; 2 c.c. of nitric acid (1.20 sp. gr.) are added, and, after boiling, 100 c.c. of water. To this solution, containing much of the tungsten as precipitated oxide, 20 c.c. of the nitric acid, and 20 c.c. of a 0.2 per cent. silver nitrate solution are added, together with 3 grams of ammonium persulphate. The mixture is gradually brought to boiling, then cooled, ferrous sulphate solution is added, and the titration completed as described for No. 1 method.

3. For the determination of manganese and chromium in a sample of steel, the manganese is determined by the method of Reddrop and Ramage. This is possible, because chromium salts in nitric acid solution are completely oxidized to chromic acid by sodium bismuthate, but the oxidation proceeds so slowly in the cold that the presence of chromium does not interfere to any appreciable extent with the determination of the manganese. After titrating the permanganate, 50 c.c. of nitric acid of 1.2 sp. gr. and 10 grams of sodium bismuthate are added, and the mixture is heated to boiling. When the oxidation of the chromium is complete with production of a clear red solution, a pinch of manganous sulphate is added, and the boiling is continued for a minute or two to decompose the permanganate formed. The small quantity of precipitated manganic oxide is filtered off, and the chromic acid in the filtrate is determined in the usual manner by means of ferrous sulphate and potassium permanganate, as in method No. 1.

Electrolytic Precipitation of Chromium.[2]—H. R. Carveth and B. E. Curry directed their efforts to the electrolytic production of chromium from chromic acid. Little was known as to the conditions under which metallic chromium might be deposited in the electrolytes of chromic acid solution.

[1] *Engineering*, Oct. 27, 1905.
[2] *Journal of Physical Chemistry*, 1905.

Previous results show that, with an impure acid, chromium was deposited, this occurring instantaneously at 18 deg. C. for a current density of about 80 amperes per sq. dcm. With pure acid the deposition was also obtained, but not so readily as with the impure acid; the decomposition voltage was found to be 2.31 volts. The effect of sulphuric acid was an increase in the deposition of metal, and in some cases more than one-half the total chromium was removed as metal. In all cases the solution was colored brown, and chromic salts were produced; a brown precipitate was also formed at the cathode. It was found that the electrolytically deposited chromium can occlude about 250 times its volume of hydrogen, 24.6 c.c. being obtained from 0.698 gram of metal.

COAL AND COKE.

By Frederick Hobart.

The coal production of the United States, which reached a total of 351,187,209 short tons in 1904, was still larger in 1905, the total being 388,772,167 short tons, a gain of 10.7 per cent. The production by States is given in the accompanying table. The year 1905 was generally a prosperous one for coal miners. Demand for coal was large, and the talk of over-production, which was heard so much in 1904 and in the earlier part of the year

PRODUCTION OF COAL IN THE UNITED STATES.

(In tons of 2000 lb.)

States.	1904.			1905.		
	Short Tons.	Value at Mines.		Short Tons.	Value at Mines.	
		Total.	Per ton.		Total.	Per ton.
BITUMINOUS.						
Alabama	11,273,151	$14,091,439	$1.25	11,900,153	$14,875,191	$1.25
Arkansas	2,009,451	2,612,286	1.30	(e) 2,000,000	2,600,000	1.30
California	79,062	376,494	4.75	48,558	130,065	2.68
Colorado	6,721,147	8,401,434	1.25	8,844,711	11,498,124	1.30
Georgia and N. Carolina	400,191	480,229	1.20	385,600	482,000	1.25
Illinois (f)	37,077,897	40,774,223	1.10	37,183,374	38,689,858	1.04
Indiana	9,872,404	10,859,644	1.10	9,772,404	10,261,024	1.05
Indian Territory (f)	3,320,057	6,375,453	1.92	2,970,961	5,398,589	1.82
Iowa	6,542,005	9,813,007	1.50	6,728,000	10,460,000	1.55
Kansas	6,322,875	8,852,025	1.40	6,780,225	10,509,349	1.55
Kentucky	7,167,324	7,122,563	0.99	8,038,646	7,810,154	0.97
Maryland	4,277,196	5,346,495	1.25	4,855,928	6,312,706	1.30
Michigan	1,414,834	2,286,160	1.62	1,380,307	2,199,207	1.59
Missouri	4,115,695	6,749,381	1.64	4,733,164	7,809,721	1.65
Montana	1,359,409	2,039,113	1.50	(e) 1,500,000	2,325,000	1.55
New Mexico	(f) 1,613,334	2,484,534	1.54	1,576,000	2,364,000	1.50
North Dakota	269,297	336,621	1.25	(e) 300,000	375,000	1.25
Ohio	24,583,815	27,042,197	1.10	25,834,657	27,643,083	1.07
Oregon	111,540	278,850	2.50	(e) 110,000	275,000	2.50
Pennsylvania	99,660,167	108,564,182	1.09	119,361,514	127,534,895	1.07
Tennessee	4,782,302	5.977.877	1.25	5,195,200	6,753,760	1.30
Texas	1,195,944	2,033,105	1.70	(e) 1,200,000	1,920,000	1.60
Utah	1,563.274	2,345,061	1.50	1,594,943	2,392,415	1.50
Virginia	3,576,092	6,515,770	1.85	4,113,950	7,610,808	1.85
Washington	2,905,689	5,520,809	1.90	2,818,042	5,495,182	1.95
West Virginia	30,222,881	30,222,881	1.00	35,283,382	30,557,938	0.87
Wyoming	4,996,828	8,744,449	1.75	5,446,525	9,531,419	1 75
Alaska and Nevada	78,868	236,604	3.00	85,000	297,500	3.50
Total Bituminous	277,512,729	$326,482,886	$1.18	310,040,644	$354,011,988	$1.14
ANTHRACITE.						
Colorado	55,404	166,212	3.00	60,503	181,509	3.00
New Mexico	(f) 24,707	78,074	3.16	24,000	78,000	3.25
Pennsylvania	73,594,369	161,907,612	2.20	78,647,020	178,528,735	2.27
Total Anthracite	73,674,480	$162,151,898	$2.20	78,731,523	$178,788,244	$2.27
Total Coal { Sh. Tons	351,187,209	488,634,784	1.39	388,772,167	532,800,232	1.37
Total Coal { Met. Tons	318,681,678	....	1.53	352,694,110	....	1.52

(e) Estimated. (f) Fiscal year ending June 30.

just closed, has gradually died out, though there was some harassing competition in the Middle West. The increased production was generally absorbed by a consumption which was without parallel in previous years. In the last quarter of the year mining was interfered with to a considerable extent by difficulties in railroad transportation. There was not equipment enough to move the coal offered, and this difficulty was sufficient to restrict mining in many places. This was the greatest trouble of the year.

General Review of the Industry.

Pennsylvania continues to lead the coal mining industry of the country. Producing practically all the anthracite and 30 per cent. of the bituminous, its mines furnished about 49 per cent. of the total coal output. Following it, but at a long interval, the important coal producers in order were Illinois, West Virginia and Ohio, mining respectively 9.8, 8.4 and 6.7 per cent. of the total. West Virginia is growing faster than any other State, and the probabilities are that, in two or three years at most, it will pass Illinois and take second place in the list.

The year was marked by the opening of new mines in different fields, but to a greater extent by the introduction of improvements at many mines; largely in the way of the introduction of mechanical haulage, the use of electricity for transmitting power, the installation of washing plants and generally in the application of labor-saving appliances and the improvement of the marketed product. The use of coal-cutting machines made less progress in 1905 than in some previous years. This was largely due to the conditions of the Interstate Agreement, which covers mining in the important districts of Western Pennsylvania, Ohio, Indiana and Illinois. The differential between pick-mining and machine-mining allowed by the contract is not sufficient to make the introduction of machines profitable, in most cases; so that operators have hesitated to make the investment required for their introduction. The greatest progress in machine-mining was made in West Virginia, where the agreement was not in force.

The production and shipments of anthracite coal in 1905 were the largest on record; they are given in detail further on in this article.

The shipments of bituminous coal to the seaboard for consumption were, probably, the largest on record, and the demand for coal throughout the Eastern territory was very large. Conditions in the Central West also favored a large output.

The shipment of coal up the Lakes, which supplies a large and growing section of the Northwest, increased in a much smaller proportion than had been expected, notwithstanding the advantages of a long season of navigation. The total shipments reported from Lake Erie ports for the season of 1905 were 10,574,198 tons; as the total for 1904 was 9,297,480 tons, the

increase shown was only 1,276,718 tons, or 13.7 per cent.; a considerable gain but still a disappointment to shippers. The coal tonnages reported as passing through the Sault St. Marie canals—that is the portion of the tonnage which goes to ports on Lake Superior—was as follows:

	1904.	1905.	Changes.
Anthracite	991,228	984,701	D. 6,527
Bituminous	5,463,641	5,524,355	I. 60,714
Total	6,454,869	5,509,056	I. 54,187

The total increase was only 0.8%, though the use of coal for mines, railroads and other purposes in the Lake Superior district must have showed a much larger gain. In part the small gain was due to the difficulty in securing cargoes promptly toward the end of the season. In part also it was due to the increasing quantity of Illinois coal which is delivered by rail at the receiving ports on Lake Michigan. These shipments began early in the year, when extra supplies were needed on account of the late movement on the Lakes. They continued through the season, introducing a new element into the coal trade of the Northwest.

Accidents.—The accident record of the year was not a good one, and a number of explosions and other disastrous calamities have been recorded. It looks very much as if attention had been so much concentrated in increasing production that precautions for safety had been measurably neglected.

Changes and Consolidations.—In the anthracite region there was a still further concentration of interests, by the sale of two large independent producers. The more important was that of Coxe Brothers & Co., to the Lehigh Valley Railroad. The late Eckley B. Coxe, a man of great resource and resolution, consolidated a group of collieries around Drifton and built a railroad—the Delaware, Susquehanna & Schuylkill—to serve them. After a long and bitter fight, he forced the recognition of his company as one of the corporations entitled to a share in the trade, and to independent representation in the markets; and he secured a contract under which his company ran its own locomotives and trains to tidewater over the Lehigh Valley tracks. This arrangement continued after his death; but last year his heirs sold the stock of the company to the Lehigh Valley. The price paid was about $20,000,000. This was about at the rate of $10 per ton of yearly output. This sale leaves only one large company—the Lehigh Coal and Navigation Company—not directly under the control of the great transportation companies.

The number of the less important independent operators was reduced by the sale of the Jermyn estate to the Erie Railroad Company for about $2,000,000. This leaves G. B. Markle & Co. almost the only large colliery

owners outside of the big companies. A number of purchases of scattered coal tracts in the Wyoming valley were made late in the year, including some lands still unworked. These have been consolidated principally in the hands of two companies—the Schuylkill Coal and Iron Company and the Shankferoth Coal Company—the real ownership of which is not clear. The report most generally credited is that these operations are in the interest of the Delaware & Hudson Company.

In the western bituminous field there were many changes, all in the direction of combination and consolidation. In the Pittsburgh district these are noted elsewhere. In Indiana several combinations have been found which include a number of smaller companies, the most important being the Vandalia Coal Company. The same condition exists in Southern Illinois, where a number of small companies have been brought into three large concerns. In both Indiana and Illinois the new combinations are either directly under railroad control, or have intimate relations with the railroads, the evident object being to put the conduct of the business as much in the hands of the railroads as possible.

In the central and northern field of Illinois there were many changes of ownership, here also the tendency being to consolidation. The railroad control, however, is not so clearly manifest in those fields.

In Ohio there were not many changes. In West Virginia there was a good deal of buying of undeveloped coal lands. Some of these were bought for immediate opening, but a large proportion will be held in reserve, at least for a time. An important area of coal territory in West Virginia will be opened by the construction of the Deepwater-Tidewater railroad, the western end of which is now well advanced toward completion, and will furnish an outlet for many new mines. The eastern end of this new road will nearly parallel the Norfolk & Western for many miles, and the road will have its tidewater terminus on Hampton Roads.

West of the Mississippi few changes are to be reported. In the coalfields of New Mexico and the Indian Territory there has been some consolidation, partly in railroad interests, and partly in that of one or two important copper companies in Arizona, who want to control their own coal and coke supplies.

Labor Conditions.—The coalfields have been measurably free from strikes and labor troubles. Some disputes have arisen in Kentucky and Tennessee and in the central district of Pennsylvania, but these have been generally settled by agreement. Some trouble threatened from the passage of the Illinois law requiring the employment of shot-firers. A question arose over the payment of these men, but it was finally settled by arbitration.

Since the great strike of 1902, the anthracite mines have been operated under the compromise award made by the Anthracite Strike Commission. The bituminous coalfields of Pennsylvania and the mines of Ohio, Indiana

and Illinois have been under what is generally known as the Interstate agreement of 1904. Both of these agreements were to expire by their own limitations on April 1, 1906. At the close of the year preparations were being made by both miners and operators for the negotiation of new agreements, the miners generally looking for an increase of wages and other concessions, and the operators opposing any changes.

The strike of the United Mine Workers in Alabama, which began in 1904, still nominally continues. The large furnace companies, however, seem to have been successful in establishing the "open shop," and the production of the State showed a fair increase.

Coal Ownership by Railroads.—A decision rendered by the United States Supreme Court in an appeal taken from the Interstate Commerce Commission has been widely taken as prohibiting railroad companies from dealing in the coal which they transport. It does not, however, bear this interpretation. The decision was, in effect, that a railroad company cannot discriminate in its own favor, but must charge itself with the same rates for the same service as it charges other shippers. In other words, a railroad company transporting coal from mines which it owns cannot use the double character of owner and carrier to undersell other shippers. The decision seems to be one difficult of enforcement.

The most important ownership of coal mines by carriers is found, of course, in the anthracite region of Pennsylvania. Here substantially all the coal going to market is owned by the carriers. The output of the comparatively few independent collieries is practically bought by the railroad companies, or their controlled organizations, on an agreed scale, which has been for some time about 60 per cent. of the selling price at tidewater. Most of the anthracite companies are protected by old charters, which antedate any provisions of the State law or constitution prohibiting railroad ownership; and, therefore, cannot be touched in the exercise of their dual functions.

Colliery Operation.—Changes during 1905 were not especially marked, and no important improvements were made. Progress continued on the lines of electrical transmission of power; the substitution of mechanical haulage for animal power in mines; improved tipples and other appliances for shipping coal. In haulage, the electric motor, the compressed air locomotive and wire rope traction systems are all coming into more extended use, the electric motor apparently finding most favor. There has been no advance in ventilating systems. In fact, the accident list of the year shows that there is too much tendency to push the extension of mining operations in advance of provisions for proper ventilation. In the anthracite country there has been an advance in breaker construction and machinery, in several new breakers. Among the features of these is the substitution of mechanical slate-pickers for the old system of hand picking.

In some large collieries, the substitution of water-hoists for pumps has proved a success in handling large quantities of water at low cost.

The coal-testing plant established in St. Louis during the Louisiana Purchase Exposition has been continued in operation, under the direction of Edward W. Parker and J. A. Holmes, of the United States Geological Survey. The results of the first year's operations have been embodied in an elaborate report, issued by the Survey in April, 1906.

Production by States.

The production of coal by States is given in the accompanying table. We give below some notes of operations in various States.

Alabama.—Chief Mine Inspector J. M. Gray reports the coal production for the year 1905 as follows, in short tons:

	Tons.	Per ct.
Lump	1,022,379	8.5
Nut	437,719	3.7
Slack	1,345,217	11.3
Run-of-mine	8,986,838	75.6
Small mines	108,000	0.9
Total	11,900,153	100.0

The total in 1904 was 11,273,151 tons, showing an increase in 1905 of 627,002 tons, or 5.6 per cent. The more important producing counties were: Jefferson, 5,816,164 tons; Walker, 2,819,073; Bibb, 1,381,675; Tuscaloosa, 852,249 tons.

The inspector's report on coke shows that the State produced 2,756,698 tons of coke against 2,284,095 tons in 1904. There are 9982 coke ovens in the State, with 647 of them not producing during the year. Jefferson county made out of this total amount 2,334,618 tons of coke.

Alaska.—Some production has been made of local importance, but systematic operations are still in the future. The coal resources of the territory are important, as shown in a paper by Alfred H. Brooks, published in the *Engineering and Mining Journal*, Sept. 2, 1905. There are already known deposits of anthracite, bituminous coal and lignite. Mr. Brooks claims that the "coalfields now known in Alaska are about equal in area to those of Pennsylvania, but it is probable that the Alaskan will be found to have many times the area of the Pennsylvanian."

Arkansas.—The coal of the Carboniferous coalfield of Arkansas is of a higher grade than any mined west of the Mississippi. Practically the whole of the Arkansas coal product comes from this field. About one-half of this product is classed as semi-anthracite, which burns with a short, hot, smokeless flame, similar to that of anthracite. Unlike anthracite, however, this coal is easily broken up, and in mining and handling makes a large amount (about 30 per cent.) of slack or fine coal, the greater part of

which is wasted or sold at prices below the cost of production. The entire product is consumed at present by Kansas City packers. Experiments made at St. Louis show that this anthracitic slack makes a high grade of briquets, and is also an excellent gas producer.

Colorado.—The production in this State shows a considerable gain, owing to extended operations by the Colorado Fuel and Iron Company, and to the opening of new coalfields, chiefly in Routt county.

Illinois.—According to the report of the Bureau of Labor, coal mined during the fiscal year ending June 30, 1905, amounted to 37,183,374 short tons, valued at $38,689,858, an increase of 105,477 tons in quantity, but a decrease of $2,084,365 in value, as compared with the preceding fiscal year. Distinguishing between those commercial, or shipping, mines and those that are worked intermittently to supply a local demand only, the State's output was contributed thus:

	No.	Tons mined.	Average.
Commercial mines	397	35,956,543	90,571
Small mines	593	1,226,831	2,069
Total	990	37,183,374	92,640

The large mines are thus seen to have supplied 96.7 per cent. of the whole output, or 0.2 per cent. more than their share in the previous year. Commercial mines increased in number by 17, and small ones by 41 over the previous year; both classes showed diminished average outputs. The disposition of the product was this:

	Tons.	Per Cent.
Sold and shipped from mines	31,667,073	85.2
Supplied to locomotives	1,178,237	3.2
Sold to local trade	2,600,808	7.0
Used, or wasted, at mines	1,737,256	4.6

The amounts, and the average value per ton of the grades into which the output was sized, were:

	Tons.	Per Cent.	Value.
Run of mine	9,248,558	24.9	$1.062
Lump	16,819,321	45.2	1.291
Egg	1,716,219	4.6	1.237
Nut	2,036,152	5.5	0.865
Pea	6,247,511	16.9	0.480
Slack	1,115,613	2.9	0.301
Total	37,183,374	100.0	$1.041

The use of mining machines is growing. The tonnage of coal undercut by machinery was 8,202,066, or 22 per cent. of the whole, as against 20 per cent. cut in this way during the preceding year. The average

price per long ton paid for pick mining was 57.82c. and for machine mining, 44.32c. The total number of miners employed was 41,202, assisted by 12,234 men and boys underground, and 5794 on the surface.

Fatalities numbered 199, a rate of 3.4 deaths per 1000 men employed, as compared with a rate of 2.87 during the previous year. Accidents involving the loss of a month or more time happened to 535 men, or to nine out of every 1000 employees.

Indiana.—The general business in the coal-mining districts of Indiana for 1905 show a decided decline as compared with 1904 and 1903. The selling price of coal was higher, but during the summer months the mines were operated perhaps less than an average of three days per week. During the fall and winter months the market was keen, but the car shortage provoking. A significant feature of the business in Indiana during the year was the merging of 35 mining companies into eight district merger companies with large capital. There were 25 new mines opened and 8 mines abandoned, making a net gain of 17 new operating shafts.

The total number of mine employees was 18,450. There was an increase of about 1000 inside men, but a decrease of about 400 outside employees; the latter, chiefly on account of consolidation of shipping mines. The production decreased about 100,000 tons, as compared with 1904. The causes assigned for this decrease are heavy spring floods, lack of summer market, and car shortage during the last quarter.

The amount of money spent in improvements is about the same as for 1904, when it was $74,231. The amount of money expended in the purchase of mines and coal land by the merger and other companies will run up into at least two figures in the million-dollar column. Very little coke is produced in Indiana, although extensive preparation is said to have been made for coke-ovens in the southern Indiana field. There were fewer fatalities and a smaller number of serious and minor accidents to employees and to mining property in 1905 than in 1904.

Indian Territory.—The report of William Cameron, United States

	1903–04.	1904–05.
Mines in operation	117	109
Coal produced, tons	3,320,057	2,970,961
Value of coal production	$6,375,453	$5,398,589
Value per ton, run-of-mine	$1,923	$1,818
Coke ovens in blast	286	186
Coke produced, tons	50,210	41,193
Men and boys, underground	7,194	6,575
Men and boys, above ground	1,066	1,062
Fatal accidents	30	44
Non-fatal accidents	69	70
Lives lost per 1000 employees	3.6	5.9

Mine Inspector, for the fiscal year ending June 30, 1905, shows a decrease of 349,096 tons in output when compared with that of the previous fiscal

year. No particular reason for the decrease can be pointed out, other than that petroleum is generally supplanting coal for railroad and manufacturing power in the region in which Indian Territory coal is consumed. Statistics of the coal industry for the past two years compare as above.

During the year 1904–05 there were 22 workings abandoned and 14 new mines were opened; the abandoned ones were nearly all small workings, while the new openings are mostly substantial producers, so that the total productive capacity of Indian Territory is about double the amount of last year's output. In past years mining has been confined to the limits of the Choctaw nation; under more recent developments the Creek and Cherokee nations are increasing in importance.

Iowa.—Very little change occurred in the coal production, or in coal mining conditions, in this State.

Kansas.—Coal mining in this State has found some advantage in the gradual diminution of supplies of natural gas, and in the great activity in zinc mining in the southeast counties and the adjoining section of Missouri.

Kentucky.—There was great activity in the purchase of coal lands and the organization of new companies, especially in the eastern section of the State. The results will doubtless be more apparent in 1906 than they were in 1905.

Maryland.—The coalfield of this State is in the Upper Potomac Basin, which also includes a part of West Virginia. The region is served by the Cumberland & Pennsylvania Railroad; the report of that road gives the total coal mined in the Cumberland region in 1905 at 6,226,284 tons. Of the amount mined last year 2,096,213 tons were produced by the Consolidation Coal Company; 1,370,356 tons by the Davis Coal and Coke Company; 787,829 tons by the Black-Sheridan-Wilson Company; 244,684 tons by the Maryland Coal Company; 231,581 tons by the George's Creek Coal and Iron Company; 224,423 tons by the Piedmont & George's Creek Coal Company; 218,544 tons by the American Coal Company; 119,689 tons by the New Central Coal Company; 143,900 tons by the Cumberland Coal Company (Douglas mine), and the balance by the other companies operating in this and the Potomac Basin. In round numbers 4,197,440 tons were mined in the George's Creek region, and 2,028,844 tons were mined in the Potomac Basin. The disposition of the coal mined was as follows:

Shipped by Balt. & Ohio R. R.	4,309,071
Shipped by Penna. R. R.	938,679
Shipped by Ches. & Ohio Canal	163,847
Local consumption and at mines	814,687
Total	6,226,284

The Baltimore and Ohio Railroad carried about 70 per cent. of the coal mined.

Michigan.—The notable feature of the year was the incorporation of the Consolidated Coal Company, with $5,000,000 capital stock, to take over a number of coal-mining companies in Michigan. The properties merged are: Saginaw Coal Company, Pere Marquette Coal Company, Shiawassee Coal Company, Barnard Coal Company, Central Coal Company, Wolverine Coal Company, Uncle Henry Coal Company, Standard Mining Company, Cass River Coal Company, Riverside Coal Company, Northern Coal and Transportation Company, Chappell & Fordney Company. The office of the new company is at Saginaw. The total capacity of the mines controlled is about 1,200,000 tons annually and the company will sink shafts for more mines. The officers of the company are: President, Walter E. Eddy; vice-president, Harry T. Wickes; secretary-treasurer, G. L. Humphrey. Of the mines owned by the 12 companies nine are located in Saginaw county and three in Bay county.

Missouri.—A considerable increase is reported in this State; but mining was carried on without special features or changes.

Montana. (By Capt. J. B. Lucas.)—Seven counties in Montana, and 44 properties, furnish the coal for commercial use and railroad consumption; while there are some 36 properties, or holes in the ground, which are either idle or practically non-producing mines. Fortunately, the coal of Montana is so widely distributed and so easily mined, in most cases, that the farmer in the eastern portion of the State, where the supply of timber is limited, can easily in a day or so lay in a supply of fuel, generally a good quality of lignite coal, sufficient to supply his needs for heating and other domestic purposes for 12 months. In all, Montana possesses an area of 1300 square miles of known coal-bearing formation, ranking ninth as compared with the other States in this regard. It must be borne in mind that until 1888 there was no commercial output of coal, practically no coal mined. Coal was discovered in Gallatin county, eight miles east of Bozeman, in 1867, and the mine opened was the first producer in the State. It is located on the line of the Northern Pacific, to which company it now belongs. A washing plant of large capacity is now in course of construction. Most of the larger coal operations in the State are owned or controlled by the large copper companies of the Butte district.

New Mexico.—The report of J. E. Sheridan, mine inspector of New Mexico, for the fiscal year ending June 30, 1905, shows that the total production of coal in the Territory was as follows, in short tons:

	Tons.	Per Cent.
Coal sold	1,472,102	95.9
Used at mines	62,196	4.1
Total mined	1,534,298	100.0

The coal sold shows a decrease of 122,482 tons, or 7.7 per cent., as com-

pared with the previous year. The falling off was largely due to lack of railroad transportation in the first half of the year, resulting from bad weather and serious washouts on the railroads, by which all freight traffic was delayed. In the second half of the year there was an increase in business.

The total number of persons employed at the mines was 2132, of whom 1660 worked underground, and 472 on the surface. Of these 52 underground and 37 on the surface were classed as boys. The average coal mined per employee was 720 tons per year.

The number of employees killed during the year was five, an average of 2.345 per 1000 employed. Four of these were killed by falls of coal or rock; one in uncoupling a mine car underground.

The production of coke for the year was 76,737 short tons, an increase of 40,937 tons over the previous year. Of the coke made in 1905, the Dawson Fuel Company reported 49,075 tons. This company has 125 ovens at Dawson, and is building 200 more. The Raton Coal and Coke Company made 27,662 tons of coke; it has 84 ovens and is building 80 more, at Blossburg.

Mine Inspector Sheridan refers to several projects for developing the coal areas of the Territory, especially the coking coal of Colfax county. Several of these include ownership of coal lands as well as railroad. They are largely in the interests of the great copper companies of Arizona and northern Mexico.

Ohio.—The coal industry in Ohio was probably more affected by competition than that of any other State in 1905. The increase shown in 1905, as compared with the previous year, was moderate only.

Pennsylvania.—This State, which produced about half the coal mined in the United States last year, showed an increase of 10 per cent. in output, as compared with 1904. The anthracite production is referred to elsewhere. The gain in the Pittsburgh district and other sections of the western bituminous district was everywhere large.

Tennessee.—In this State, as in Kentucky, there has been much activity in the purchase of coal lands and the opening of new mines.

Washington.—The only official report of coal mining in Washington is

	Output.	Average Days Operated.	Employees.
Kittitas	1,279,636	244	2,443
King	1,083,785	240	1,701
Pierce	453,121	260	841
Lewis	1,500	200	2
Total	2,818,042	236	4,987

published biennially, the latest one having covered the years 1903–04;

D. C. Botting, State Mine Inspector, has, however, courteously afforded us statistics covering the year 1905. A few small operations remain to be heard from, but the totals are reasonably complete.

The statement by counties is shown on the preceding page.

This shows an average output per man of 564 tons per year, or 2.4 tons per day worked. The number of men classed as miners is not known.

The output of coke was 51,072 tons, all from Pierce county, an increase of 4897 tons over the output of 1904. The production of coal shows but little change from the previous year.

West Virginia.—(By J. W. Paul, Chief Mine Inspector.)—In 1905 no particular novelty characterized the coal industry of the State; there was the usual increase in tonnage, and also the clamor of the operators for railroad cars in which to ship the product. The trade, viewed from the operators' side, was not productive of much profit, by reason of the low price for coal during the greater part of the year. Much complaint was heard, during the latter half of the year, of the scarcity of cars; but this condition seems to have been the result of a general demand upon the railroads for cars for all purposes. Again, the increase in the production of coal in the fields now developed was quite large, and the growth was in excess of the ability of the railroads to move the tonnage offered.

Based upon the production for the fiscal year, ending June 30, 1905, the production of coal for the calendar year 1905 reached 35,000,000 net tons; and of coke, about 2,800,000 tons. That the railroads took care of an increased tonnage, a table given herewith attests; the figures are in tons:

	1900.	1903.	1904.	1905.
B.&O.R.R.Company .	4,832,853	5,749,829	7,988,955	7,864,308
C.&O.Ry.Company...	4,116,970	3,609,467	5,976,644	7,700,811
N.&W.Ry.Company. .	4.181,400	6,075,934	6,749,131	8,227,419
K.&M.Ry.Company. .	713,811	†1,161,457	1,503,861	1,865,130
W.Va.C.&P.Ry.Co. . .	1,729,568	1,568,158	1,809,833	1,821,690
Totals..........	15,574,602	18,182,845	24,033,424	27,479,358

†Estimated.

During the past two years, improvements were in progress which placed the railroads in a position to accommodate the growing traffic; but the lack of motive power and coal cars was keenly felt. The trade names of West Virginia coals are becoming so firmly fixed and so widely known in the general markets that an ever increasing demand is made upon the producers.

In the past, the most enthusiastic predictions of future production were surpassed; and, on account of the numerous opportunities offered by the State, it becomes mere guesswork to hazard a prediction upon the future tonnage.

The year 1905 marked an important era in railroad construction within the State; notably, the completion of the Coal and Coke Railway, connecting Charleston with Elkins; the grading of many miles of the Deepwater

Railway; the opening of the Kanawha & West Virginia Railroad Company's line along Blue Creek; the extension of the Coal River & Western Railroad; and the grading of the extension of the Morgantown & Kingwood Railroad.

Since 1896 this State has been third in rank among the coal-producing States, a position it now holds, being surpassed only by Pennsylvania and Illinois.

The consolidations of consequence during the year consisted of the absorption, by the Fairmont Coal Company, of the Pittsburgh & Fairmont Fuel Company and the Southern Coal and Transportation Company; by the Sunday Creek Company, of the Kanawha & Hocking Coal and Coke Company; and by interests allied with the Berwind-White Coal Mining Company, of the W. P. Rend Mines.

During the last part of the year an effort was made to consolidate, under one ownership, about 28 mining companies on Cabin Creek in Kanawha county; also a similar effort was made to consolidate, under another ownership, all the mines on Paint Creek, Kanawha county; but at the close of the year neither effort had been consummated, though the indications are favorable to an early consolidation of these fields.

During 1905 there were no strikes of any consequence in any part of the State, aside from local troubles of short duration, in which the mines in Mason county were the most affected.

The Coal Markets.

The following notes relate to the leading coal markets of the country. They have a direct bearing on the course of coal production for the year:

Birmingham, Ala. (By L.W. Friedmann.)—With a strike among several thousand union coal miners on at some of the larger collieries in the State, the production of coal in Alabama for the year 1905 shows an increase as compared to 1904. There was steady operation throughout the year at the commercial coal mines, where union labor is employed, at the convict mines and at the non-union mines, while at the mines where new labor has been employed in the place of strikers, the production increased right along until it approached the normal conditions.

Some important moves were made in the coalfields of Alabama during 1905. In addition to the starting of developments in the upper Cahaba coalfields, in the northeastern part of Jefferson county, and in St. Clair county, two large mines being in operation already, a large number of coal companies were organized and work begun on new mines in different portions of Jefferson county, in Walker, Shelby, St. Clair and other counties. The Pratt Consolidated Coal Company, now with a daily output of 8000 tons of coal, purchased in 1905 many thousand acres of coal lands in Jefferson and Walker counties, and mines are being opened thereon. Just

as soon as railroad extensions can be constructed to the new mines the coal production will commence and will be pushed until this company is getting out 20,000 tons of coal a day. The Tennessee Coal, Iron and Railroad Company began work on a shaft near the Pratt mines, and it is proposed to use this shaft outlet for the coal from three of the old mines in addition to working the coal that is struck in its development. A complete electric haulage system is placed. The Alabama Consolidated Coal and Iron Company also began boring for coal under a seam of coal now being worked at the Lewisburg mines, in Jefferson county. If this plan should be successful, there is a belief that other companies will make a similar test, the geologists and other experts all expressing the belief that in Alabama there are two seams, one over the other. These are but a few of the important events which took place in the coal industry in Alabama during 1905. The purchase of coal property on all sides was almost continuous.

There was a steady demand for all the coal mined. With but a few weeks' exception there was no railroad car shortage in this State. The railroads in the district give the coal industry much attention, and without exception every railway line ordered new coal cars, which began arriving just as the coal demand picked up and there was urgent desire to move the product promptly.

Chicago. (By E. Morrison.)—With the steady growth of population and manufacturing industries, the coal trade of Chicago for 1905 is satisfactory to everybody connected with it, as regards volume of business. More coal was sold and used than in any previous year. Prices, however, were for the greater part of the year so low as to make profits small both to the miner of the coal and the dealer.

This condition is a result of the great anthracite strike of 1902–3, when the need of fuel to take the place of anthracite and the high prices of anthracite caused hundreds if not thousands of new mines in the Middle West to be opened. Once opened the mines were considered of value for working purposes, regardless of the condition of the market. With coal under almost any farm in Illinois, combination of the bituminous field owners was impossible. Hence there was over-production and little profit for anybody in the business.

In the last quarter of 1905, however, the situation was somewhat better for the dealer on account of the car shortage, which restricted shipments to the local market. But if this has aided the dealer and the individual operator, it has been at the expense of the producers. The mines of Indiana and Illinois were not operated for more than one-third to one-half time, because of the lack of cars.

Fewer contracts were made in 1905 for supplies of coal than last year, the tendency of users of coal being to hold off from contracts in view of the open markets being so favorable to them. In general, there was more

buying for immediate delivery, outside of contracts. These conditions made the market fluctuating to a greater degree than in 1904.

At the end of the year, Western bituminous averaged \$2.25@2.75 for lump and egg; \$2@2.30 for run-of-mine and \$1.50@1.60 for screenings. These prices were in general advances over the first of the year, for in January lump and egg brought only \$1.80@2; run-of-mine, \$1.50@1.70 and screenings, \$1@1.40.

Eastern coals had a somewhat parallel course to Western in demand and transportation conditions. During the first half of the year the market was depressed for these coals; in the last quarter they commanded premiums over previous prices and were hard to get. Hocking at the end of the year brought \$3@3.75, in January it was quoted at \$2.75@3, against circular prices of \$3.20 at the beginning and \$3.40 at the end of the year. Smokeless coals gained in popular favor, unquestionably, through the anti-smoke crusade and the growth of appreciation of high-grade fuels, and advance in price through the car shortage. Youghiogheny was sold mostly on contracts, but advanced on free sales 30@40c. during the year.

The anthracite market suffered from the general prosperity of the country. There was a reluctance of consumers to lay in their winter stocks, in the spring and summer, through a feeling that the 10c. a month discounts did not amount to much, and the popular belief that there would be plenty of coal for everybody when it should be needed. Retailers expressed these opinions to wholesalers, and generally delayed laying in coal that the consumer would not buy immediately.

Receipts of coal by lake at Chicago and Waukegan were 960,250 tons anthracite and 78,455 tons bituminous, making a total of 1,038,705 tons, against 1,024,853 tons in 1904. There was no division of receipts in 1904 as to anthracite and bituminous by the custom-house officers, who say, however, that bituminous in 1904 was about 10 per cent. of the total, and was chiefly Hocking coal.

Cleveland and the Lake Trade. (By G. H. Cushing.)—During the past two years the coal business has grown complex, due in the main to the unprecedented increase in the productive capacity. At the outset it is to be said that while the lake season of navigation in 1905 was fully two and a half months longer than during the preceding year, the movement by lake does not show the gain which might have been expected. The statement of one of the railroad general superintendents is representative. He says that during the season of 1905 the lake-dock facilities have been engaged only to 30 per cent. of their normal capacity. This is the more striking when it is said that coal-dock facilities on the South Shore of Lake Erie did not undergo the same process of reconstruction and improvement as did the ore docks. In fact, the railroads showed a very slight inclination to increase their coal-handling facilities.

The causes leading up to this situation are numerous. It was apparent that at the opening of the season of navigation the docks in the upper lake region had more than their usual supply of coal on hand. At the same time there was a revulsion of feeling in the Northwest against the high prices which had been paid for the Pennsylvania and Ohio coal, which seemed entirely out of keeping with the prices in the home market, the cost of transportation and the reasonable profits to the middlemen for the handling of this fuel. This feeling was not appeased by a quarrel which arose between two of the principal shippers. The firm of M. A. Hanna & Company, which had sold its mines to the Pittsburgh Coal Company, had a five-year contract for the handling of the output of those mines. This expired with the beginning of April, 1905, and was not renewed. An open rupture was declared, each threatening to cut into the territory of the other.

This feud appeared at the time when Northwestern consumers were about to come into the market for the purchase of their season's supply of coal. A price war broke out on fuel coal to lake steamers only, but the hope was aroused in the Northwest that sooner or later prices of coal generally would be reduced by the competition. Having a large supply of coal on hand, the consumers and dealers of the Northwest could easily afford to await a better time for buying their supply, which they did.

About that time reports began to appear all through this territory that the western mines were beginning to sell to the trade in the Northwest, taking away from Ohio and Pennsylvania their best customers. This dates back to the time of the anthracite coal strike, when the bituminous deposits of the East found a ready market near at home, and when the supply was diverted from the upper lakes to meet the demands of the East. Seizing the opportunity, the Illinois mines began to find an opening for themselves in the Northwest, which they have not since been willing to yield. When the dispute between the two big shippers in question arose, producing hesitancy in the minds of the northwestern buyers, the western mines were quick to take advantage of the situation; and they were aided by favorable railroad rates.

Under these conditions, prices early in the year were weak, run-of-mine steam coal selling at 90 @ 95c. at mine in Ohio. Consumption was heavy, however, and there was a gradual improvement. Moreover there was a decrease in coal supplies, owing to difficulties in transportation.

The relation between the supply of railroad cars and the condition of the crops is very close. An abundant crop of farm products almost invariably means a shortage of equipment, provided the supply of rolling stock before was not out of proportion to the demand of the trade—a condition which has never existed. Toward the end of the year the abundance of the crop having been established, this shortage of cars was evident in an exaggerated form, and produced an immediate effect upon the condition of the coal

trade in this territory. The demand for cars was so urgent that the railroads were forced to divert equipment from the lake trade six weeks before the close of the season of navigation. This put an end virtually to lake shipments, long before the shippers were ready. At the same time railroad operations were so heavy that they were using coal in excess of their contract requirements. This caused them to confiscate coal consigned to others.

At the same time it had the effect of shortening the supply at the commercial centers and caused competitive bidding for the amount of coal actually being shipped. As a result, prices ran up violently at the close of the year. The last three months found the price increased from 95c. to $1 at the mines to $1.40 to $1.50 at the mines, with possibilities of still higher prices. Consumers were actively bidding one against the other at the close of the season and the result was striking.

Pittsburgh. (By S. F. Luty.)—The production of coal in the Pittsburgh district showed a considerable increase, and the demand was also large during the greater part of 1905. But, while business was good, the profits were not satisfactory or as large as 1904.

Prices throughout the year were unusually low. At the opening, quotations were based on run-of-mine coal at $1.05 a ton at the mine, but this rate was invariably shaded when a good contract was offered. In July as low as 95c. a ton was quoted, but some sales were reported to have been made at 80c. a ton. There was no improvement in prices until October, when rates were fixed as follows: Run-of-mine, $1.20 a ton; $\frac{3}{4}$-in., $1.30; $1\frac{1}{4}$-in., $1.40; slack, 90c. A shortage of railroad cars and an increased demand made it possible to obtain these prices, and in November an advance of 10c. a ton was announced by some of the principal interests, but it was not maintained, owing to unusually mild weather and favorable transportation facilities. The year closed with the market in fairly good condition and a favorable outlook for the new year.

Some important deals were made by the Pittsburgh Coal Company, the leading producer. The first was in April, when it entered into a contract with the United States Steel Corporation to furnish coal to its plants in the Pittsburgh district for a period of 25 years. It is estimated that fully 3,000,000 tons will be required annually. The National Mining Company, a subsidiary interest of the Steel Corporation, was more active in 1905, and its production amounted to about 800,000 tons. In November the Pittsburgh Coal Company closed a contract with the Republic Iron and Steel Company for its coal requirements for a period of five years. The tonnage will run from 500,000 to 750,000 tons annually. At the same time the company renewed its contracts for furnishing coal to the St. Louis and Cincinnati gas works.

The Jones & Laughlin Steel Company has become the largest independ-

ent coal producing concern in the district and is probably the largest in the country, its production for 1905 amounting to fully 2,500,000 tons. This company, however, is not a competitor in the market, as it consumes its entire product. In June it bought 9600 acres of valuable coal land, increasing its holdings to 15,000 acres.

Several independent coal companies added to their holdings and increased their capacity, notably the United Coal Company, its production in 1905 being almost double that of the previous year. In February it bought two large mines from the Hazel Kirk Coal Company. The production of the Pittsburgh Buffalo company, J. W. Ellsworth & Company, and the Washington Coal and Coke Company also was greatly increased. In July the Carnegie Coal Company absorbed the Chartiers Coal and Coke Company, which added to its strength.

As the mining rate had been fixed for a period of two years in 1904, there was no uncertainty, and contracts were made early. Competition for trade was very keen and some contracts were made at very low prices. One of the remarkable features of the year was the absence of labor troubles. While a few local strikes occurred, none amounted to anything. Some foreigners attempted to cause a suspension at the mines of the Ellsworth company for a trifling dispute over the discharge of one of the employees, but the leaders of the United Mine Workers prevented it and insisted upon the strict observance of the contract. Another feature of the year was the successful shipments of coal by river to the lower markets, there being comparatively few accidents. The rivers were navigable for several days every month in the year, and the shipments exceeded those of 1904. Owing to the large stocks constantly on hand at all lower ports, prices were not satisfactory to the shippers. Fully 16,000,000 bushels went out in January and about 13,000,000 bushels in March. The next largest shipments were made in June and October, 12,000,000 bushels being shipped in each of these months.

The report of the Pittsburgh Coal Company, the leading operator of the district, for the year 1905 shows that the coal mined was as follows:

	1904.	1905.	Changes.
Pittsburgh dis.	12,783,067	12,926,954	I. 143,887
Hocking dis.	1,349,428	1,371,120	I. 21,692
Total coal	14,132,495	14,298,074	I. 165,579
Coke made	206,005	355,873	I. 149 868

The financial statement shows $32,000,000 preferred and $32,000,000 common stock. The general mortgage bonds outstanding are $24,093,000; in addition to which there are $1,896,341 bonds of subsidiary companies and $1,500,000 notes issued for purchase of Monongahela River Consolidated stock.

The earnings of the company and its subsidiaries—not including the Monongahela River Company—for 1905 were as follows:

Net earnings	$3,255,358
Depletion of coal lands	$ 593,075
Depreciation of plant	830,271
Interest on first mortgage bonds	1,207,087
Dividends	343,273
Total deductions	$2,973,706
Bal. appropriated for working capital	$ 281,652

The amount charged to dividends above was two-thirds of dividend No. 22 on the preferred stock, the other third having been provided out of the earnings of 1904. Dividend No. 23, which accrued during the year, was not paid.

The Seaboard Bituminous Trade. (By E. K. Judd.)—The conditions in the bituminous coal market on the seaboard have been largely modified in recent years by the practical understanding on rates reached by the railroads, and with the gradual absorption or suppression of the smaller producers.

A large proportion of the bituminous coal marketed on the seaboard is sold on yearly contracts, April 1 being the usual date for the beginning of the contract year. This leaves only the smaller part of the trade open to fluctuations throughout the year. At the opening of 1905, however, there was considerable speculative selling, owing to the severity of the winter, which tied up the railroads and the coastwise vessels, making deliveries at several points impossible, and preventing contract deliveries on time everywhere. Trade was generally good except during July and August; in the last third of the year it was exceptionally active.

At the approach of April, the beginning of the contract year, it became an object of concern to learn what action the main-line railroads would take in arranging their through rates for the season, and to learn what would be the consensus of opinion among the miners. Hesitation was removed by the announcement of the railroads that last year's rates would continue for another year. The Norfolk & Western and the Chesapeake & Ohio made a rate of $1.35 to Norfolk and Newport News; the Pennsylvania, with a haul of 100 miles shorter distance, made the rate $1.50 to New York city. The mine workers of central Pennsylvania, at their convention in Altoona, showed a conciliatory spirit, but their desire for position at the opening of the following year prevented them from noting the fact that their high wages constituted an insurmountable handicap in competing with the chief production of the non-union territory. They demanded and obtained a continuation of the 62c. rate for pick mining. The first contract for the year was greeted with relief, when it was noted the previous year's price was maintained. Subsequent contracts not only confirmed this continu-

ation, but, if anything, showed a tendency toward higher prices, particularly on the better grades. The average basis for most of the year's contracts was \$2.70 f. o. b. New York harbor shipping points, or \$1.15 @ \$1.20 at the mines. For the ordinary grades of coal, a slightly lower price was accepted in the hope of creating a more stable basis; on the poorer grades, 5 or 10c. was sometimes conceded in order to get a contract, particularly in new territory. These concessions had the effect of delaying the placing of some contracts, a tendency which the insistence on high prices for the better grades only tended to confirm.

Trade in New York harbor on the whole was active throughout the year. In the spring, when arrivals were irregular, a certain amount of speculative business crept in. The railroads lent a hand in correcting this by keeping a strict oversight on the unloading of coal from their cars at terminal points. Whenever any shipper allowed a number of loaded cars to accumulate, his requisition for empties at the other end of the line was discounted; one or two applications of this remedy was sufficient to instill into the mind of the shipper the necessity for unloading his coal promptly. This feature, while it prevented speculation, at the same time had a depressing influence upon market prices. During the summer trade was rather quiet, although no stagnation was apparent at any time. With the approach of cold weather the market resumed activity in an aggravated form.

A notable feature of the market during the year was the remarkable persistency with which New England, particularly the far eastern and shoal-water ports, laid in supplies of coal. Demand from this territory was unusually active throughout the year. Contracts, too, began to play an important part in the far eastern trade, greatly surpassing in amount the purchases of coal in the open markets. So keen were the New England manufacturers to lay by supplies of coal that, at times, the unloading facilities of the receiving ports were taxed to the utmost, and demurrage charges on vessels kept waiting were not uncommon. By the end of the year contracts had been disproportionately fulfilled, leading to the belief that before the end of the contract season other supplies would have to be sought.

Trade along the north shore of Long Island Sound presented a spectacle never before seen. In the early spring, as soon as the ice-bound harbors were open, a rush of coal set in which swamped the unloading facilities of the ports; the New York, New Haven & Hartford railroad was especially unable to cope with the demands upon its facilities. By April this rush had subsided and then the district set itself diligently to work to lay in supplies of anthracite at the prevailing low prices of that month. They gave their entire attention to this kind of coal, and bituminous trade suffered accordingly. This was all the more to be regretted for the reason that freight rates to this territory by water were never so low, and the procras-

tination in laying in supplies of soft coal was dearly paid for later on when boat rates advanced 25c. per ton. When, in August, the demand for hard coal had been nearly satisfied, bituminous coal was sought with an urgency out of all proportion to the possibility of supplying it. From that time until the end of the year the supply of soft coal could not begin to keep up with the demand, while advancing the freight rates and the impossibility of getting supplies by rail made it a costly experience.

All-rail trade throughout the year was as active as the producers would permit it to become. By the middle of September, the delay attending the return of cars from New England roads led the Baltimore & Ohio and the Reading railroads to embargo shipments in their cars to the New York, New Haven & Hartford. Demands for all-rail shipments continuing, the producers were led to adopt the expedient of adding to the price of such coal, and a basis of $1.50 was adopted for the purpose of checking all-rail shipments.

Producers complained almost continuously throughout the year of insufficient car supplies, the shortage at some times being almost acute. On the whole, however, those producers who showed reasonable speed in unloading their cargoes at terminal points were fairly well supplied with cars. Ordinary shortage was not so objectionable as the irregularities that marked the supply in certain districts; this condition made it impossible for shippers to depend upon a smooth running of their mines. Railroad transportation during the year was unusually good; at certain seasons of the year better than schedule time was maintained for long periods.

For carrying coal along the Atlantic seaboard, vessels were in good supply throughout the year. With the opening of the Delaware river and Chesapeake bay, in the middle of March, the large number of the boats waiting for cargoes had a depressing influence on freight rates, and many of them were driven to accept lumber or anthracite coal, which paid better rates. As soon as any congestion was noticeable at harbor points the boat owners were not slow to take advantage of it by demanding the loading and discharging clause, by the provision of which they were able to get their cargoes promptly, regardless of the trouble and expense to which they may have put the shippers in providing the cargo at the dock. During the summer, freight rates to New England fell to an abnormally low figure. With the departure of many small boats to other lines of trade, a shortage set in and when demands for them arose in the early fall, rates quickly recovered to a high figure. Large boats were in abundant supply throughout the year; only the small ones were sometimes hard to get.

The year was free from labor disturbances of any importance. Operators in union districts, however, are beginning to feel the burden under which they are laboring, as compared with their competitors in non-union fields.

The Anthracite Trade.—The anthracite market—which is mainly an

Eastern market—showed in 1905 the results of centralized control. Not only was the largest annual output of anthracite ever produced in this country marketed without the slightest fluctuation from established prices, but its monthly quotas were forthcoming with the regularity of any well-established manufacturing enterprise. The realization of this latter condition in the trade, advantageous alike to miner, transporter, and consumer, has resulted from the four years' operation of the retrograding summer discount scheme, through which, by common consent of the operators, a reduction of 50c. per ton on prepared sizes takes effect on April 1, to be restored by monthly increments of 10c. until it disappears in September. For two years now, as a result of this plan, April, May and June have seen the heaviest shipments of the year—a complete inversion of the course of shipments in earlier years, and accomplished in spite of the preponderating importance of domestic anthracite for winter fuel.

The output for anthracite, as shown below, was the largest ever reported; yet this tremendous output was attained without special effort. During August, many mines were shut for days at a time; shipments suffered from exceptionally bad weather early in the year, and from insufficient car supplies during its closing months, and at no time were the selling agencies besieged with buyers.

New York harbor trade in the domestic sizes recorded no noteworthy events, its ups and downs having been synchronous with the fluctuation of the thermometer. The 50c. discount on April 1, even though magnified by the local dealers to a concession of 65c. per ton on domestic sizes, created no enthusiasm, although dealers had no difficulty in disposing of their consignments promptly upon arrival. Ominous reports of growing dissatisfaction with their terms of employment, among the anthracite miners, and the suspicion that the prevailing wages agreement would not be replaced at its termination, April 1, 1906, without some friction led prudent householders to stock their bins early, and this demand, coupled with a simultaneous car shortage, protracted the spring activity well into summer. During the warm months, domestic trade was exceedingly dull, and not even the approach of winter prices afforded any stimulus. It was not until the last of October that demand improved and a month later before it became urgent, when a car shortage assisted in creating a condition of some stress.

The market for steam sizes, pea, buckwheat, rice and barley, is influenced by two circumstances, unknown to the domestic sizes: A large part of the small coal is recovered at the breakers in the preparation of the larger domestic sizes, so that its output varies directly with that of the prepared sizes and cannot be controlled separately. Additional sources are found in the old culm piles now treated by washeries; then, in the market, the fine sizes come into direct competition with bituminous coal for use in steam

plants and for this reason are unable to establish prices on the basis of their own supply and demand. Neither are they subject to the summer discount plan adopted for the domestic sizes. Ordinances for the prevention of smoke in New York and other cities give anthracite steam sizes a slight advantage. During 1905, the small sizes maintained their selling price fairly constant; a slight flurry occurred in March, but it was quickly subdued. All through the summer they came forward in abundance, but only in June did any dealers find it expedient to shade prices. Late in September an active demand arose for pea coal; many large steam plants filled their storage room and arranged for additional space. This soon developed into an urgent inquiry for all the fine coals, even washery coal moving off quickly; prices, however, did not respond.

Transportation was excellent throughout the year except for a few days late in January, when a blizzard tied up the coal industry at both ends and along the line. The railroads, however, were the first to recover, and during the rest of the winter it was the unloading of frozen coal, the ferrying from New Jersey terminal points to New York, and the towing of barges by the outside route to Boston that made trouble. Shortage of cars was a fertile source of complaint throughout the year. In view, however, of the enormous tonnage moved last year, it might be suspected that some of the complaint came from force of habit rather than from real hardships. The anthracite trade twice a year suffers a periodical shortage of cars—in the spring, when the opening of Lake shipping causes a demand for cars to the Lake ports, and in the autumn, when the grain harvests of the West call for all available cars. Then, too, the shipping of anthracite to New England by all-rail routes, a recent innovation, has grown marvelously, and already the main-line roads are considering means to suppress the exodus of their cars to this territory.

New England has an acute recollection of its distress during the winter of 1902, and with the first note of discord in the anthracite field started in to lay up supplies. In its haste to amass hard coal, it nearly neglected to provide for its bituminous supply. Indeed, along the Sound, it was not until August, when anthracite had nearly resumed its normal winter price, that bituminous coal was able to attract any attention, and to depose hard coal from its monopoly of the market. The Western anthracite markets have never fully recovered from the blow received during the strike of 1902. In the large cities, where the chief market for this coal is found, the use of bituminous coal received a great impetus in that year. The higher-grade coals of West Virginia and the Hocking Valley found a place which they have largely retained in domestic use. They have replaced a certain part of the anthracite formerly used, and in so far limited its consumption. Making this allowance, the Western trade was generally steady and even, very much as the Eastern market was. The Lake trade, owing to the longer sea-

son of navigation, was better distributed, and there was not the rush and confusion which marked the opening and close of the season of 1904. Prices were generally maintained, as in the East, and the trade showed no special incidents.

The transfer of the properties of Coxe Brothers & Company and the Jermyn estate, with other changes in holdings, have been already referred to. No further attempt was made to change the status or ownership of the Lehigh Coal and Navigation Company. Under the control of the New York, New Haven & Hartford Company, the New York, Ontario & Western continued to work in harmony with the other companies.

The condition of the mine workmen during the year has been good. Local disturbances have occurred, but have usually subsided promptly under the influence of the Conciliation Board. This organization has grown into high favor with both operators and employees, and with its three years' experience has become a valuable feature. Wages continued to be adjusted on the sliding scale, under the supervision of Commissioner Neil. They remained unchanged during the early months, but in the latter half rose as much as 6 per cent. in some months. The approach of the termination of the present agreement, which expired April 1, 1906, caused a certain amount of uneasiness. In preparation for it a convention was called at Shamokin, on Dec. 14, and it was strongly urged upon the local unions to exercise caution in the selection of their delegates. As a result, the convention was marked by conservatism, and the policy to be adopted in future arrangements with the operators was left to the discretion of the Mine Workers' president and executive board.

At the mines, some difficulty arose at the prospect of enforcing a new State statute covering the employment of child labor in anthracite mines. The miners' union has been instrumental in obtaining this legislation, which prohibited the employment of boys under 14 years old in breakers, and under 16 years in mines. This provision was generally complied with, although it raised the minimum age by two years in each case. Another provision required a testimonial as to a certain amount of elementary school education from all employed under 21 years of age. This clause led to a certain amount of hardship and was declared unconstitutional by the County Court.

Another decision, by the Pennsylvania Superior Court, declared unconstitutional that part of the miner's certificate law demanding two years' experience in a Pennsylvania mine as a requisite to the holding of a certificate, without which no miner was permitted to work underground. This decision removed one of the weapons which the union miners had relied on in case of a contest with their employers, since no restriction now prohibits the giving of the necessary certificate to a qualified miner from another State.

Anthracite Shipments.—Shipments of anthracite by companies for two years were as follows:

	1904.		1905.		Changes.
	Tons.	Per ct.	Tons.	Per ct.	Tons.
Reading	11,399,362	19.8	12,574,502	20.5	I. 1,174,880
Lehigh Valley	9,611,426	16.7	10,072,120	16.4	I. 460,694
N. J. Central	7,201,276	12.5	7,983,274	13.0	I. 781,998
Lackawanna	9,333,069	16.3	9,554,046	15.6	I. 220,977
Del. & Hudson	5,276,797	9.2	5,640,628	9.2	I. 363,731
Pennsylvania	4,765,963	8.3	4,890,635	8.0	I. 124,682
Erie	5,711,173	9.9	6,225,622	10.1	I. 514,449
N. Y., Ont. & West	2,646,730	4.6	2,864,096	4.6	I. 217,366
Del., Sus. & Schyl	1,546,476	2.7	1,605,378	2.6	I. 58,902
Total	57,492,522	100.0	61,410,291	100.0	I. 3.917,679

The total increase in 1905 was 6.7 per cent., the shipments in 1905 being the largest on record. Every company showed an increased tonnage, though all did not partake equally of the total increase. None of the changes is striking; it was known that the Reading had been accelerating its coal production, and that the Lackawanna has had to curtail the output of several of its largest collieries pending the installation of improved mine plants and breakers.

The production of anthracite in Pennsylvania in 1905 has been ascertained as below, the shipments of anthracite being always reported in long tons:

Shipments, as above	61,410,291
Used in operating mines and by employees and railroads	8,810,263
Total mined, long tons	70,220,554
Total mined, short tons	78,647,020

The proportion allowed for use at mines and breakers, and by employees, was 9.1 per cent. for the year, as shown by the Pennsylvania reports. Of the coal shipped to market about 60 per cent. is of the larger sizes, known as prepared sizes, which are used for household purposes and are sold at full prices. The remaining 38 per cent. is of the small or steam sizes, which come to a considerable extent, in combination with bituminous coal, and are sold at lower prices than the prepared sizes. The coal used at collieries is almost entirely of the small sizes. Included in the shipment is a certain quality of what is called washery coal; that is, small coal obtained by washing the old culm banks which abound in the anthracite region. For several years this has been from 3 to 4 per cent. of the total, but it has a tendency to decrease as the old banks, which accumulated when the saving of coal was not so close as it is now, are gradually exhausted. The actual output of washery coal in 1905 was 2,644,045 tons; a decrease of 156,421 tons from the previous year.

Foreign Coal Trade.—The foreign coal trade of the United States is

unimportant, as compared to its production. The exports are less than 3 per cent. and the imports 0.5 per cent. of the home production.

Exports of coal and coke from the United States for the year are reported by the Bureau of Statistics of the Department of Commerce and Labor as below, in tons:

	1904.	1905.	Changes.
Anthracite	2,228,392	2,229,983	I. 1,591
Bituminous	6,345,126	6,959,265	I. 614,139
Total coal	8,573,518	9,189,248	I. 615,730
Coke	523,090	599,054	I. 75,964
Total	9,096,608	9,788,302	I. 691,694

The coke went to Mexico and Canada; the coal exports were distributed as follows:

	1904.	1905.	Changes.
Canada	6,577,954	6,964,630	I. 386,676
Mexico	880,747	927,170	I. 46,423
Cuba	519,227	564,385	I. 45,158
Other W. Indies	253,585	300,776	I. 47,191
France	10,948	4,554	D. 6,394
Italy	69,930	68,369	D. 1,561
Other Europe	63,476	28,354	D. 35,122
Other countries	197,651	331,010	I. 133,359
Total	8,573,518	9,189,248	I. 615,730

The coal to other countries goes chiefly to South America. The exports to Canada in detail were:

	1904.	1905.	Changes.
Anthracite	2,193,746	2,187,450	D. 6,926
Bituminous	4,384,208	4,777,180	I. 392,972
Total	6,577,954	6,964,630	I. 386,676

In 1905 Canada took 98.1 per cent. of the anthracite and 75.8 per cent. of all the coal exported.

Imports of coal and coke into the United States for the year are reported as follows:

	1904.	1905.	Changes.
Canada	1,211,304	1,331,292	I. 119,988
Mexico	221	40	D. 181
Great Britain	135,292	94,600	D. 40,692
Other Europe	601	395	D. 206
Australia	235,069	184,426	D. 50,643
Japan	45,429	41,956	D. 3,473
Other countries	759	134	D. 625
Total coal	1,628,675	1,652,843	I. 24,168
Coke	....	181,376	....

Coke was not reported separately in 1904. With the exception of a small quantity from Germany it comes from British Columbia. Most of the

coal from Canada is also from British Columbia and is delivered on the Pacific coast; but some of it is Nova Scotia coal, which comes chiefly to Boston. Of the coal imported in 1905 there were 34,262 tons classed as anthracite.

The Bureau of Statistics reports bunker coal—that is, coal supplied to steam vessels—at various ports in 1905 as follows: Atlantic, 16,338,994 tons; Gulf, 2,087,538; Pacific, 2,736,405; Lakes, 7,408,253; total, 28,571,-190 tons. This is a decrease of 274,261 tons from the previous year.

Production of Coal in the World.

The gain in coal production is steady in most of the world's coalfields, and the total approached closely—if it did not quite reach—900,000,000 tons last year. The production is given in the following table:

COAL PRODUCTION IN THE CHIEF COUNTRIES OF THE WORLD. (In metric tons.)

Countries.	1901.	1902.	1903.	1904.	1905.
Asia:					
India	6,741,899	7,543,272	7,557,400	7,682,319	7,921,000
Japan	8,945,939	9,701,682	10,088,845	11,600,000	11,895,000
Australasia:					
New South Wales	6,063,921	6,037,083	6,456,524	6,116,126	6,035,250
New Zealand	1,259,521	1,386,881	1,442,916	1,562,443	1,415,000
Other Australia	926,188	931,148	771,536	769,723	805,000
Europe:					
Austria Hungary (c)	40,746,704	39,479,560	40,160,823	40,334,681	40,725,000
Belgium	22,213,410	22,877,470	23,913,240	23,380,025	21,844,200
France	32,325,302	29,997,470	34,906,418	34,502,289	36,048,264
Germany (c)	153,019,414	150,600,214	162,457,253	169,448,272	173,663,774
Italy	425,614	413,810	346,887	359,456	307,500
Russia	16,526,636	16,465,836	(f)17,500,000	19,318,000	17,120,000
Spain (c)	2,651,857	2,807,550	2,800,843	3,123,540	3,199,911
Sweden	271,509	304,733	320,390	320,984	331,500
United Kingdom	222,614,981	230,728,563	233,419,821	236,147,125	239,888,928
North America:					
Canada—					
Western	1,861,248	1,826,221	1,791,798	2,619,816	3,183,909
Eastern	3,788,168	4,699,396	4,700,645	4,194,939	4,775,802
United States	266,078,668	273,600,961	317,272,110	318,275,920	352,694,110
South Africa (a)	1,388,205	2,213,275	2,957,736	3,015,000	3,218,500
All Other Countries (e)	2,500,000	3,500,000	4,000,000	4,250,000	4,550,000
Totals	790,349,184	804,115,125	882,865,185	867,020,658	929,622,648

(a) Transvaal, Natal and Cape of Good Hope. (c) Includes lignite. (e) Estimated. (f) Estimated by Minister of Finance.

Some details in relation to foreign coal production are given herewith:

Belgium.—The output of coal in 1905 was 21,844,200 metric tons; a decrease of 6.6 per cent. from 1904. This was due in large part to a general strike of the miners early in the year. The foreign trade was as follows:

Years.	Exports.		Imports.	
	1904.	1905.	1904.	1905.
Coal	5,067,037	4,681,489	3,701,240	4,227,028
Coke	879,883	977,086	338,127	359,404
Briquets	539,364	483,135	45,600	70,605
Totals	6,486,284	6,141,710	4,084,967	4,657,037

The excess of coal exports over imports in 1905 was 454,461 tons; of coke, 617,682 tons.

Canada.—The production of coal in 1905 was 8,775,933 short tons (7,959,771 metric tons). This output came from the old mines of Nova Scotia in the East and of Vancouver Island, British Columbia, in the West; from the newer mines of the Crow's Nest district in the Kooteney region of British Columbia, and in Alberta; Nova Scotia produced about 60 per cent., British Columbia 25, and Alberta, 15 per cent.

France.—The total coal production of France in 1904 and 1905 is given in the following official figures. in metric tons:

	1904.	1905.		1904.	1905.
Nord and Pas-de-Calais	21,718,269	23,166,981	Bourbonnais	998,278	1,041,571
Loire	3,598,591	3,744,228	All other districts	1,595,304	1,635,096
Bourgogne and Nivernais	1,975,322	1,970,827	Lignite	665,572	701,034
Gard	1,833,247	1,946,272			
Tarn and Aveyron	1,783,383	1,842,255	Totals	34,167,966	36,048,264

This shows an increase of 1,880,298 tons, or 5.5 per cent., in the total output.

The foreign coal trade for the two years was as follows:

	Exports.		Imports.	
	1904.	1905.	1904.	1905.
Coal	1,120,153	1,658,680	10,884,468	10,513,920
Coke	160,580	242,040	1,656,361	1,632,710
Briquets	66,788	88,940	528,107	399,390
Total	1,347,521	1,989,660	13,068,936	12,546,020

Included in the exports in 1905 are 132,940 tons of coal and 32,470 tons of briquets furnished to steamships.

Important new discoveries by test-boring are reported at Abancourt and Pont-a-Mousson, in French Lorraine.

Germany.—The coal production of Germany is reported as follows, in metric tons:

	1904.	1905.	Changes.		1904.	1905.	Changes.
Coal	120,948,050	121,190,249	I 242,199	Coke made	12,331,163	16,358,324	I. 4,027,161
Brown coal	48,500,222	52,473,525	I. 3,973,303	Briquets made	11,413,000	13,000,000	I. 1,587,000
Total mined	169,448,272	173,663,774	I. 4,215,502				

The foreign movement of coal was as follows:

	1904.	1905.	Changes.
Imports	7,299,042	9,399,693	I. 2,100,651
Exports	17,996,726	18,156,998	I. 160,272

The changes in 1905 were chiefly due to the interruption of output in the important Rhenish-Westphalian district, early in the year.

Spain.—The following table gives the production of coal and coke in Spain for the past two years, in metric tons:

	1904.	1905.	Changes.
Anthracite	165,008	159,372	D. 5,636
Bituminous	2,707,629	2,885,711	I. 178,082
Lignite	103,272	154,828	I. 51,556
Total mined	2,975,909	3,199,911	I. 224,002
Coke made	435,318	466,689	I. 31,371
Briquets made	307,479	297,371	D. 10,108

Spain's foreign trade in fuel, for the same years, amounted to:

	1904.	1905.	Changes.
Imports:			
Coal	2,129,893	2,206,398	I. 76,505
Coke	177,085	145,288	D. 31,797
Exports:			
Coal and coke	2,767	2,171	D. 596

Most of the imports are from Great Britain.

United Kingdom.—The production of coal in 1905 showed only a small variation from the previous year. The total was 236,111,150 long tons (239,888,928 metric tons). This was an increase of 3,699,366 long tons, or 1.6 per cent., over 1904.

Exports of fuel with coal sent abroad for the use of steamships engaged in foreign trade were as follows for the full year, in long tons:

	1904.	1905.	Changes.
Coal	46,255,547	47,476,707	I. 1,221,160
Coke	756,949	774,110	I. 17,161
Briquets	1,237,784	1,108,445	D. 129,329
Total exports	48,250,280	49,359,272	I. 1,108,992
Steamer coal	17,190,900	17,396,146	I. 205,246
Total	65,441,180	66,755,418	I. 1,314,238

There was an increase last year of 2 per cent. in the coal consumed beyond the limits of the United Kingdom, this total being about 28 per cent. of the coal mined. There are practically no imports of fuel.

Coke Production.

The accompanying table (on page 109) gives the production of coke in the United States in 1904 and 1905 by States.

The production and consumption of coke depends almost entirely on the condition of the metallurgical industries. In a year of great activity in the iron and steel trade, it was inevitable that there should be a l arge increase in coke production, and this is shown by the table.

Pennsylvania and West Virginia are the larger producers, and both States are favored by the possession of important fields of coal of coking quality. The largest and best known of these is the Connellsville field

PRODUCTION OF COKE IN THE UNITED STATES.
(In tons of 2000 lbs.)

States.	1904.			1905.		
		Value.			Value.	
	Short Tons.	Total.	Per Ton.	Short Tons.	Total.	Per Ton.
Alabama	2,284,095	$5,550,351	$2.43	2,756,698	$6,753,910	$2.45
Colorado and Utah	789,060	2,461,867	3.12	795,650	2,585,863	3.25
Georgia and N. Carolina	75,812	235,017	3.10	69,200	204,140	2.95
Indian Territory (*a*)	44,808	205,669	4.59	41,193	189,488	4.60
Kansas	9,460	28,853	3.05	10,200	31,110	3.05
Kentucky	64,112	153,869	2.40	65,475	166,961	2.55
Missouri	2,446	6,482	2.65	3,100	8,525	2.75
Montana	41,497	208,935	6.77	43,500	282,750	6.50
New Mexico (*a*)	35,800	102,030	2.85	76,737	222,537	2.90
Ohio	109,284	333,316	3.05	112,250	342,363	3.05
Pennsylvania	13,281,475	30,680,207	2.31	18,519,310	45,187,116	2.44
Tennessee	379,240	1,133,927	2.99	382,300	1,166,015	3.05
Virginia	1,101,716	2,478,861	2.25	1,203,650	2,837,578	2.35
Washington	46,175	215,176	4.66	51,072	240,038	4.70
West Virginia (*a*)	2,276,451	5,645,598	2.48	2,738,777	6,846,942	2.50
Other States	1,464,130	4,978,042	3.40	1,535,000	5,219,000	3.40
Total Coke. Short tons	22,005,561	$54,418,200	$2.47	28,404,112	$72,284,336	$2.55
Total Coke. Metric tons	19,967,841	....	2.72	25,768,210	....	2.81

(*a*) Fiscal year ending June 30.

in the southwestern part of Pennsylvania. Another important coking field is found in Alabama. The Eastern coke is used entirely by the blast furnaces, steel mills and foundries; that made in the Southwest and in Montana is used mainly by the copper and lead smelters.

The production given does not include gas coke, which is a by-product in the manufacture of gas for city supply. This is used to a small extent by foundries, but is chiefly sold in the cities where it is made, as household fuel.

The coke made in the United States continues to be made chiefly in the old beehive ovens. The use of the retort or by-product oven increases slowly, though those plants already in service have been successful everywhere.

COPPER.

By W. R. Ingalls.

The production of copper by domestic mines in the United States in 1905 was 871,634,245 lb., against 817,715,005 lb. in 1904. The details of these statistics, showing in what States the increase occurred, are given in the following tables. These statistics are based on reports received directly from the producers of Lake and blister copper, except in the cases of Colorado and Idaho, the figures for Colorado being checked by our own reports, for which the statistics of the State officials have been adopted.

The actual production of copper in the United States is much greater than the production of the domestic mines, a large quantity of ore, matte and pig copper being imported for refining in this country. The total production of American refiners in 1905 was about 1,025,000,000 lb., of which 219,000,000 lb. were marketed as Lake copper (a comparatively small portion of which was electrolytically refined), about 760,000,000 lb. as electrolytic copper, and the remainder either as casting or as pig copper.

A distinction should be made, however, between the standard brands of Lake copper and the arsenical copper which is produced by the South Range mines. The production of the South Range mines in 1905 was 43,542,334 lb. The relative proportions of Lake and electrolytic copper for several years, so far as statistics bearing upon this important subject are available, are shown in the following table:

PRODUCTION OF COPPER ACCORDING TO CLASS.

(In pounds.)

Year.	Total Domestic.	Total Foreign.	Grand Total.	Lake.	Electrolytic.
1897	501,370,295	26,938,254	528,308,549	145,839,749	250,000,000
1898	535,900,232	36,055,352	571,955,584	156,669,098	314,107,776
1899	581,319,091	40,659,868	621,978,959	155,845,786	386,410,356
1900	600,832,505	62,484,290	663,316,795	144,227,340	466,092,663
1905	871,000,000	191,000,000	1,062,000,000	219,000,000	760,000,000

The above table does not take into account under the caption "electrolytic" the comparatively small quantity of Lake copper which is refined in that way by the Quincy and Calumet & Hecla companies. They refine thus only selected mineral, especially high in silver, the separation of which more than offsets the slight deterioration in grade which is suffered by the conversion of "Lake" into electrolytic.

There are in the United States eight important electrolytic refineries (exclusive of the two which refine Lake copper) of which the largest are situated in the vicinity of New York. These are the works of the Nichols Copper Company, at Laurel Hill, L. I., the De Lamar works, at Carteret, N. J., the Raritan works at Perth Amboy, N. J., the works of the American

Smelting and Refining Company, at Perth Amboy, and the Balbach works, at Newark, N. J. Outside of this district there are works at Baltimore, Great Falls, Mont., and Tacoma, Wash. The aggregate capacity of these works, at the end of 1905, was between 750,000,000 and 775,000,000 ℔. of refined copper per annum. In 1905 they were operated to full capacity. It became obvious that the further increase in the production of crude copper, which was to be expected, required additional refining capacity. This is now being provided.

A new unit has been added to the De Lamar plant, increasing its capacity 50 per cent., which is expected to be in operation by the end of April. The Raritan works are being doubled, the addition probably to be completed some time next fall. It is understood also that the capacity of the plant of the American Smelting and Refining Company will be increased. In the meanwhile, a new refinery is being erected at San Francisco by the Mountain Copper Company, and the Tacoma works are being doubled.

COPPER STATISTICS OF THE UNITED STATES.

	1902		1903		1904		1905	
	Pounds	Long Tons	Pounds	Long Tons	Pounds	Long Tons	Pounds	Long Tons
Alaska	....	....	(a)	....	2,043,586	912	4,703,600	2,100
Arizona	119,841,285	53,501	153,591,417	68,567	191,602,958	85,537	222,866,024	99,494
California	25,038,724	11,178	19,113,861	8,533	29,974,154	13,381	13,089,993	5,844
Colorado	8,463,938	3,779	7,809,920	3,487	9,401,913	4,197	9,854,176	4,399
Idaho	(a)	....	(a)	....	5,422,007	2,420	6,500,000	2,902
Michigan	170,194,996	75,979	192,299,485	85,848	208,329,248	93,004	218,999,753	97,768
Montana	266,500,000	118,973	272,555,854	121,676	298,314,804	133,176	319,179,885	142,491
New Mexico	(a)	....	(a)	....	5,368,666	2,397	5,638,842	2,517
Utah	23,939,901	10,687	38,302,602	17,100	47,062,889	21,010	51,950,789	23,192
Wyoming	(a)	....	(a)	....	3,565,629	1,592	2,393,201	1,068
Southern States	13,599,047	6,071	13,855,612	6,186	15,211,086	6,791	14,907,982	6,655
Other States	9,218,490	4,116	10,846,477	4,842	1,418,065	638	1,550,000	692
Total production.	636,796,381	284,284	708,375,228	316,239	817,715,005	365,051	871,634,245	389,122
Stock, January 1	209,587,698	93,566	162,935,439	72,739	230,111,792	102,729	208,376,672	93,025
Imports	161,551,040	72,121	167,161,720	74,626	182,292,205	81,380	210,724,685	94,074
Total supply	1,007,935,119	449,971	1,038,472,387	463,604	1,230,119,002	549,160	1,290,735,602	576,221
Deduct exports	376,298,726	167,991	312,822,627	139,653	555,638,552	248,053	548,772,403	244,988
Deduct consumption	468,700,954	209,241	495,537,968	221,222	466,103,778	208,082	612,983,199	273,652
Stock, December 31	162,935,439	72,739	230,111,792	102,729	208,376,672	93,025	128,980,000	57,581

(*a*) Included in "Other States."

The average price of Lake copper at New York in 1905 was 15.699c.; of electrolytic, 15.599. The monthly fluctuations, and comparisons with previous years, are given in the tables presented further on in this article.

Home consumption was beyond all precedent, while the foreign demand showed a considerable falling off. The output of the metal showed the effect of the extension in mining which began three and four years ago. In the Lake Superior region, in Arizona and in Montana the production of the older mines increased and a number of new mines in which work had been in progress entered the producing list.

The imports of copper into the United States in 1905 were large; the

greater part of these came from Mexico in the form of blister copper; some also came from Canada and some was received from South America, importations from the last source being on the increase. Exports to Europe fell off decidedly. A demand for about 30,000 tons from China was not quite sufficient to bring the exportation up to that of the preceding year.

The domestic consumption was far in excess of anything previously known in the United States, the average being about 51,000,000 per month. All the mills manufacturing sheet, wire, and other finished forms of copper, all the brass mills and all the manufacturers of machinery into which copper and brass enter in part, were extremely busy throughout the year. The great extension of electric work of all kinds, the construction of new electric roads and the application of electricity on existing railroads were largely responsible for the demand, which shows every sign of continuance through 1906.

The stocks estimated do not indicate that at the close of the year there were 57,580 tons of copper available for immediate delivery. As production is reported from the mines, and periods varying from 20 to 75 days for different districts are required to transport and refine the copper, there must necessarily be a large amount of copper nominally included in the stocks, irrespective of any quantities of refined metal which may remain unsold. Our estimate of this is based on the proportions of copper marketed as Lake, electrolytic, casting and pig, and the relative time required for transportation and refining. Careful inquiry has made it evident that the unsold stock of copper in marketable form at the close of the year was probably less than 5,000,000 ℔. No account is taken of stock of "mineral" at Lake smelting works, or of refined copper in consumers' warehouses. There is reason to believe, however, that the latter was, in some cases, rather large at the close of the year, wherefore the actual consumption was somewhat smaller than appears in the statistics.

Review of Copper Mining in the United States.

Alaska.—In 1905, this Territory made a large increase in its production of copper ore and furnace material, shipments of matte having been made by the Alaska Copper Company from its smelter at Coppermount, Prince of Wales Island, Southeast Alaska. This smelter is a new plant, the machinery of which is operated by water power. The furnace is of 250 tons daily capacity. The matte is shipped to the Tacoma smelter. The latter plant, as in 1904, received ores from elsewhere in Alaska, but the quantity increased materially in 1905. During the year the shipments amounted to 29,412 tons, valued at $663,506, against 12,636 tons valued at $258,302 during 1904.

It has been known for several years that there is a large area of copper-

bearing country around the headwaters of the Tanana, White and Copper rivers, Alaska, but they were not developed on account of transportation difficulties. These, however, will shortly be overcome. The first section of 35 miles of the railroad projected to open up this part of Alaska is now being constructed. Its primary object is to tap the copper deposits located on Copper river, about 90 or 100 miles above its tidewater terminus at Valdez, with the ultimate intention of extending it across country to Eagle City, on the upper Yukon, within easy reach of Dawson City. It was reported some time ago that these deposits had passed under the control of a syndicate of San Francisco and London capitalists.

Later discoveries at the headwaters of the Tanana are native copper in stringers and specks in greenstone formation. The natural outlet for these new deposits is said to be through the Copper river valley to Valdez. The Havemeyers, of New York, are reported to be interested in one group of copper properties at the headwaters of one of the tributaries of the Tanana and have a force of miners at present employed in developing them.[1]

The most recent information is to the effect that during three months, December, 1905, to February, 1906, the Alaska Smelting and Refining Company's smelting works at Hadley, Prince of Wales Island, smelted rather more than 14,000 tons of ore, from which about 800 tons of copper matte were obtained. This matte was sent to the smelter at Crofton, Vancouver Island, B. C., to be converted into blister copper[2]. The furnace at Hadley was blown in on Dec. 5. The ore was from the Brown-Alaska company's Mamie mine and the Hadley Consolidated Copper Company's Stevenstown mine, both on Prince of Wales Island, together with silicious copper ore from the Britannia mine, Howe Sound, B. C., as a flux for the local ores, which contain a large percentage of iron and insufficient silica.

Arizona.—There was a large increase in the production of this Territory in 1905, and the prospects are for a further important increase in 1906. Our statistics are based on reports received directly from all the producers, but some of these having been communicated confidentially we are unable to publish the details. The largest producer was the Copper Queen Consolidated Mining Company, which at its smelter at Douglas reduces the ore from its mines at Bisbee, and also the sulphide concentrates from Nacozari, Mexico, together with a comparatively small quantity of custom ore. The capacity of this plant is being largely increased. The Calumet & Arizona company also operates a large smelter at Douglas, whither the ore from its mines at Bisbee is transported. The Bisbee district is at present by far the most important in Arizona, being followed in order by Globe, Clifton-Morenci, and Jerome. The great increase in the output of the Old Dominion Mining Company, at Globe, was one of the important features of 1905.

[1] *Engineering and Mining Journal*, Nov. 4, 1905.
[2] This production is not included in the statistics for Alaska, no metal having been made up to the end of the year.

Recently it has been reported that sulphide ore has been struck in the lowest level of this mine, which is a highly important development, inasmuch as sulphides have heretofore had to be shipped to Globe from Bisbee. The new plant of the Arizona Smelting Company at Humboldt (near Prescott), which was under construction in 1905, will go into operation in April, 1906. This is a well constructed plant, of thoroughly modern design, and will stimulate the production of ore in the adjacent districts. The available ores being chiefly concentrates, the smelter has been equipped with reverberatory furnaces (following the large furnaces at the Washoe works, Mont.), which are to be fired with crude petroleum from California. The knowledge of Arizona copper deposits has been increased by the publication of the report by Waldemar Lindgren (U. S. Geological Survey) on the Clifton-Morenci district.

According to Mr. Lindgren the total output of the Clifton-Morenci district to the end of 1903 is estimated to be about 201,600 short tons of copper. The ores average 3 to 4 per cent. of copper as mined. By far the larger percentage of ore consists of sulphide, containing pyrite and chalcocite. The output of oxidized ore at Metcalf will continue for a number of years and possibly increase. The oxidized ore mined by the Arizona Copper Company at Metcalf is the lowest grade utilized in the district, and probably contains on an average less than 3 per cent. copper. Mr. Lindgren believes that the future ofthe camps is assured for many years, since the low-grade orebodies, which are now the mainstay of the district, are of large extent, and the production bids fair to continue, at least at the present rate, for some time to come.

Shannon Copper Company: Developments during the year ending Aug. 31, 1905, amounted to 11,931 ft. at an average cost of $3.46 per ft. Additions to reserves have more than kept pace with extraction, and the mine has never before had so much ore exposed. The output was 188,856 tons; its average content was 4.08 per cent. copper, or 0.39 per cent. less than in the previous year. Smelting ore, carrying 5.15 per cent. copper, amounted to 53,353 tons. Concentrating ore, carrying 3.66 per cent. copper, amounted to 135,503 tons. Ore from the open cuts, carrying only 2.2 per cent. copper, is mined for an iron flux, but only in quantities required by the smelter. The cost per ton for mining 188,856 tons was $2.138, the leading items being: Mining, 93.2c.; timbering, 50.1c.; tramming, 11.2c.; development and exploration, 21.7c.; filling, 5.6c.; sampling and assaying, 4.8 cents. The mill was run only two-thirds time, owing largely to interruptions in transport, due to heavy floods. Crude ore is now being screened before crushing and the fines are briquetted, for the purpose of saving excessive losses of oxidized ore in the mill; magnetic concentration and picking the ore are expected to reduce losses of these oxides in the tailing.

The furnaces put through 128,724 tons, made up of 52,926 tons of Shan-

non ore; 3519 tons of custom ore; 29,735 tons of concentrate; 3831 tons of fine ore; 12,949 tons of flue-dust; and 975 tons of copper slag. A converter plant is under construction. For converting matte, the company has heretofore depended upon its neighbors. The year's output was 11,414,271 ℔. fine copper as against 10,788,891 ℔. in the previous year. Of gold, 592 oz., and of silver, 17,127 oz. was sold. The average price received for copper was 14.24c. per pound. Operating expenses were $1,126,248 (9.87c. per ℔.), construction and development to the amount of $52,321 was charged against earnings, and selling and miscellaneous general expenses were $241,772, showing a net profit of $271,153 on the year's work. Expenditures of $45,650 were charged to capital, for permanent improvements. The company's capital is $3,000,000, and it has $480,000 of bonds.

Arizona Copper Company: The most important developments during 1905 were in the Coronado mine, where high-grade concentrating ore was opened in the 600-ft. level. A low-grade orebody 110 ft. wide was developed in the 250-ft. level in the Humboldt mine. The Clay mine shows a vein 59 ft. wide, averaging 4 per cent. copper.

The new concentrating plant, at Morenci, will be completed in May, 1906, and will have a daily capacity of 700 tons.

During the half year ending Sept. 30, 1905, 302,003 tons of first-class and of concentrating ores were extracted. Of these, 176,548 tons were obtained from Longfellow group, 67,332 tons from Metcalf group, and 58,123 tons from Coronado, the percentage of first-class and of concentrating ores being 4.47 and 95.53 respectively. The average yield of all ores was 2.38 per cent., and 43.31 tons of ore were required to produce one ton of copper.

The smelter handled 57,694 tons of ore and concentrate, and 1,209,082 ℔. of copper derived from the leaching works were smelted, resulting in a gross yield of 14,351,477 ℔. Of this 6571 tons were obtained from the smelting of copper ore and concentrate, an average yield of 11.39 per cent.

The leaching plant treated 49,408 tons of tailing, and produced 1,263,605 ℔. of copper. As compared with the previous six months, the treatment of tailing shows an increase of 8633 tons; production of copper 29 tons.

(By James Douglas.)—The copper interests in Southern Arizona were very active during 1905. The large smelter of the Copper Queen Company at Douglas, equipped as custom works, was put into more effective working condition, and an increase in size of about 50 per cent. was nearly completed. This increase, however, is intended less to augment the copper production of the works than to provide for the treatment of a larger tonnage of dry gold and silver ores. Nevertheless, the copper output of these works, which was 80,112,713 ℔. in 1905, will probably be slightly increased during 1906. The total output in 1905 of the Copper Queen mines themselves, exclusive of what was put into stock, but including

5,000,000 lb. of Copper Queen copper in the form of sulphide ore, shipped to the Globe smelter, was 69,554,322 lb.

The greatest activity prevailed in the mines known at the Bonanza group, an activity which exhibited itself in the greater production of the Calumet & Arizona smelter, which reduces the ores of that company, and is being enlarged to handle the production of ores from the other mines of the allied corporations. The production of the Calumet & Arizona smelter in 1905 was approximately 32,000,000 lb., and it may safely be assumed that this figure will be exceeded in 1906. Railway switches from the El Paso & Southwestern R. R. have been built to the Irish Mag and the Oliver shafts of the Calumet & Arizona company, to the shaft of the Calumet & Pittsburgh and to one of the shafts of the Lake Superior & Pittsburgh, all of which are now producing or prepared to produce. This is also true of the Junction.

The Clifton district slightly declined in its production, due not to any failure in the ore supply, but to interruption through floods. But nevertheless the Arizona Copper Company and the Detroit Mining Company, unless interrupted by misfortune, will turn out slightly more copper in 1906 than in any year previously, as the former company is building a new mill and the latter is doubling the capacity of its existing concentrator, and both improvements should be in operation before 1906 closes.

The larger capacity of the plants in the Clifton district is generally used to treat a lower grade of ore rather than to proportionately increase the output in copper. The concentrating ore of the Detroit Copper Mining Company in 1905 yielded only 2.09 per cent. in copper and contained no recovery values in gold and silver.

Globe also showed a large increase. As pointed out last year, the importation of sulphides from Bisbee and elsewhere enabled the Old Dominion to inaugurate matte smelting and bessemerizing with considerable economy over the old method of direct production of copper from oxidized ores. The output from the Globe smelter amounted to 28,888,959 lb., but as the company has been importing sulphides and is treating large quantities of local custom ores, this production did not come exclusively from the Old Dominion mine. The company's own mines, however, will themselves be more productive in the future, as the concentrator, which was started in August last, and which is being much enlarged, enables the company to utilize its low grade ores, which heretofore have been worthless. A check to operation was given in the early part of February, 1906, by a fire at the old Interloper shaft, but beyond curtailing production by closing temporarily the lower levels of the mine, it did no serious damage.

The same, however, cannot be said of an extraordinary flow of water on the 10th level, which followed the fire, though apparently without any assignable connection with it. Some three years ago the lower levels of the mine were flooded while opening up a rich orebody in its western

section. The disaster for the time being threatened to be almost irreparable, but the water was kept under control until recently, when it again drowned the lower levels. Radical measures, however, are being taken to handle, with much less cost through a new shaft, even a much heavier flow than has heretofore been encountered; and therefore before many months these new orebodies, which apparently exceed in both size and average richness any yet encountered, will be safely accessible.

The United Mines shipped to both the Old Dominion smelter and likewise to the Douglas smelter, and a neighboring property, the Arizona Commercial, is likewise marketing its ores. The Black Warrior and other mines in the Pinal district are yielding more than ever heretofore, and therefore the whole Globe district may be expected during 1906 to contribute 40,000,000 to 50,000,000 ℔. of copper.

The Imperial mine was a steady producer, shipping its ores to the Copper Queen smelter at Douglas; the smaller mines in the Dragoon Mountains and the Santa Ritas added to the sum total; the Yavapai country contributed, though less than it will in 1906, when the smelting works upon the Agua Fria will have yielded a year's production. The United Verde smelter runs continuously, but improvements are being made which will probably enable this very productive mine to increase its output.

No very important metallurgical modifications were made at any of the works. At the Detroit Copper Company's works, at Morenci, a reverberatory furnace was erected with a view to cleaning the slags and at the same time smelting the flue-dust. The results are not yet conclusive, but point to a favorable issue. At the Copper Queen works the larger furnaces have been designed so as to secure a much longer drop from the feed floor to the surface of the charge, the longer drop securing a very much better distribution of the ore when fed in large units from the feed cars. The older furnaces are being rebuilt on similar lines. Otherwise, no noteworthy changes in the structural plan of the works in Southern Arizona have been made, but at the Greene Consolidated Copper Company's works at Cananea very important modifications, looking to the more economical handling of the ores between the concentrator and the furnaces, will probably soon be put to the test of actual operation.

California.—The production of this State declined in 1905, because of idleness of the smelter of the Mountain Copper Company, which was enjoined on account of smoke troubles. Recently the case has been decided in favor of the company. In the meanwhile, the company has been erecting a new smelter and electrolytic refinery at San Francisco, which should go into operation in 1906.

The copper deposits of Shasta county have developed a magnitude and prospect of permanence which a few years ago was not expected, and the outlook is for a considerable increase in production from this source.

The Bully Hill company, formerly controlled by J. R. DeLamar, has been acquired by the General Electric Company. The Mammoth group of mines was purchased by the United States Mining Company (since reorganized as the United States Mining, Smelting and Refining Company), which in 1905 erected a smelting plant of capacity for 1000 tons of ore per day. This plant was completed and went into operation during the latter part of the year. It is not, as yet, equipped with a converting plant, the matte being shipped to the large smelting works of the company, near Salt Lake City, Utah, where its iron contents serve as much needed flux for the siliceous ores from Tintic, and elsewhere, which are smelted at that plant.

The Balaklava mine, in Shasta county, was purchased by the White Knob Copper Company, which had previously abandoned its mines at Mackay, Idaho, but has since passed into the hands of the Guggenheims, who are erecting a new copper smelter at Point San Bruno, near San Francisco. A large smelter for the Balaklava mine is being erected at Kennett. This will produce matte, which will be shipped to the San Bruno works.

An interesting question with respect to this new plant is the matter of copper ore supply. There are hundreds of unworked claims in California, and others that are productive; but many of them are far off in the mountains, remote from transportation facilities. There are many undeveloped copper properties, for instance, in the vicinity of Taylorsville, Plumas county; but there has been no production as yet, nor will there be until the Western Pacific Railroad, now under construction, is completed. Some ore comes from Fresno and Mariposa counties, and a little from other localities, but the total does not amount to much outside of Shasta county, where the principal mining companies have their own smelting furnaces.

The most productive copper mine in the State, except the Mountain, Mammoth, and Bully Hill, of Shasta county, is the Daisy Farm, recently opened in Placer county. This has recently been acquired by the Guggenheim interests. Ores from the Southwest, from Nevada and from Mexico will also be worked at the new plant, and, aside from ocean transportation, the lines of the Southern Pacific, Santa Fé and Western Pacific will act as feeders.

The Great Western Gold Company, operating at Redding, put its new plant into operation during 1905. The Fresno Copper Company, an English concern, continued the construction of its works, which are to have two furnaces and a converter plant. It is understood that these works, which have been designed by Wellman-Seaver-Morgan Engineering Company, will embody many new features. The property of the Trinity Copper Company was idle during the year.

Colorado.—The production of copper in this State in 1905 was about the same as in 1904. The statistics of the production, as stated in the general table previously given, are reported by the commissioner of mines for the State. The major part of the copper originating in Colo-

rado is obtained at Leadville. The copper-bearing ore from that district, and the other districts of the State, is smelted by the large silver-lead plants, the Boston & Colorado Smelting Company, at Argo, near Denver, and by one or two independent pyritic smelters.

The Golden smelter, which was largely supplied from Clear Creek and Gilpin counties, closed down during the summer, and is hardly likely to reopen under present conditions. The pyritic smelters at Silverton and Ouray were not operated in 1905, but a similar plant was built at Grand Junction, and is said to have been in operation during the year. The matte smelting furnaces of the Ohio & Colorado Smelting Company, at Salida, were operated. The Rocky Mountain smelter, at Florence, was put in operation again toward the end of the year. A matte-smelting furnace was built at Pearl, near the Wyoming line.

The discovery of good deposits of copper ore in Colorado would be extremely valuable, not merely for their copper contents, but also to serve as collecting agent for smelting purposes. In the absence of such an ore supply, attempts at matte smelting in the State do not succeed in competition with the lead smelters.

Idaho.—About 87 per cent. of the copper production of Idaho in 1905 was obtained in Shoshone county, where the Snowstorm mine is the principal producer. This mine has now been developed to a depth of over 1200 ft. by a cross-cut tunnel. The vein, which dips at an angle of about 60 deg., shows an ore-shoot over 500 ft. in length. The thinly bedded quartzite above the foot-wall is impregnated with copper carbonate to an extreme width of 60 ft. at one point above the second level, where the ore averages about 5 per cent. copper and 8 oz. silver per ton. The ore gradually fades out into the quartzite of the hanging wall.

The Seven Devils district, wherein are many copper prospects, has still failed to materialize as an important producer. The operations of the Ladd Metals Company at Landore, the central settlement of the district, have apparently proved a failure. This company purchased about 3000 tons of ore, which has been resold to the Oregon Smelting and Refining Company, for reduction at Sumpter, Oregon; a considerable quantity of this ore was delivered before the close of 1905. The ore is said to contain an average of about 12 per cent. copper, and several dollars per ton in gold and silver. It is chiefly carbonate, rather high in silica. Besides this ore, several carloads averaging upward of 20 per cent. copper, $2 gold and 11 oz. silver per ton were shipped out of the district in 1905. It is hoped that extensive developments will be undertaken in the Seven Devils during 1906.

At the Lost Packer mine, in the Loon Creek district, Custer county, a 100-ton smelting furnace was built in the autumn of 1905, but it was not expected to be ready for operation until the spring of 1906. The mine, which has been extensively developed, is said to show a body of ore, 10 to

15 ft. in width, which averages 2 to 3 per cent. copper and $5 to $7 in gold per ton. In this low-grade ore there is a streak, averaging 2 ft. in thickness, and proved for a length of 500 ft. and a depth of 400 ft., which is said to average 10 to 25 per cent. copper and 2 to 5 oz. gold per ton. The copper occurs in the form of chalcopyrite. The Lost Packer mine is 25 miles from Custer, with which it is connected by wagon-road.

The White Knob mine, at Mackay, Custer county, having been abandoned by the company, was taken over by a lessee, who put one of the furnaces in operation to work up several thousand tons of low-grade ore on hand, and produced 25 carloads of matte. The smelter was shut down late in the fall, and the leasing system at the mine extended. At present about 100 men are employed, and the results are considered promising for an increase in the output of ore in 1906. The White Knob ore deposits occur at a contact between limestone and porphyry, especially favoring the limestone, in which the ore occurs in large, irregularly shaped bodies. The contact has been opened for a depth of 700 ft. below the apex, where it is cut by an adit. Above that level the ore is carbonate; below the level sulphide ore is beginning to show. It is hoped that the mine can be made to pay under the economical management of the lessee.

In Kootenai county, the Panhandle Copper Mining and Smelting Company commenced the development of deposits of sulphide ore in the Priest Lake district, near the north end of the county. One of the deposits is said to be large, showing chalcopyrite in connection with pyrrhotite. The resources of this district have not yet been well prospected, and it is considered to be one of the best fields in the State for pioneer investigation.

Michigan.—In many respects, the year 1905 was one of the most remarkable in the history of the Lake Superior copper mining industry. Never before were the profits so large, or so many men employed and prosperity more general. Many new enterprises were launched, which give promise of bearing fruitful results, and several of the older companies are giving attention to their undeveloped lands. Production was the largest on record, in spite of labor strikes at some of the mines. The production is given in detail in the accompanying table (page 121).

(By C. E. L. Thomas.)—The largest gain in 1905 was made by the Champion, opened upon the Baltic amygdaloid bed, and owned equally by the Copper Range Consolidated and St. Mary's Mineral Land companies. It now occupies fourth place among the Lake producers, and would have made a better showing had not labor troubles interfered. Despite labor difficulties, the Baltic and Trimountain companies, also operating the Baltic lode, increased their production. Crediting the Copper Range Consolidated Company with the output of the two latter mines, which it owns outright, and one-half of the Champion's product, it turned out 32,714,859 lb. of fine copper in 1905. This company will be an increasingly important

factor in the Lake copper trade for many years, and in 1906 should make half as much copper as the Calumet & Hecla.

The Calumet & Hecla company is now shipping regularly from its Osceola mine workings, three shafts being in operation at that branch of the property, and satisfactory results are being secured. The workings on the Calumet conglomerate are furnishing practically the same production as during the last few years, and will last for 25 years more. The new mine on the Kearsarge amygdaloid is a shipper to a small extent, and production from that source promises to become larger as the openings are extended. The Calumet & Hecla is opening a new mine at the Delaware, in Keweenaw

COPPER PRODUCTION IN MICHIGAN.
(Pounds of fine copper.)

Mines.	1900	1901	1902	1903	1904	1905
Adventure	....	....	606,211	2,182,608	1,380,480	1,606,208
Ahmeek	....	....	....	....	350,000	1,552,957
Allouez	....	....	....	....	....	1,167,957
Atlantic	4,930,149	4,666,889	4,949,368	5,505,598	5,321,859	4,049,731
Baltic	1,735,060	2,641,432	6,284,819	10,580,997	12,177,729	14,384,684
Cal. & Hecla	81,403,041	82,519,676	81,248,739	76,490,869	80,341,019	83,812,370
Centennial	....	806,400	....	....	641,294	1,446,584
Champion	....	....	4,165,784	10,564,147	12,212,954	15,707,427
Franklin	3,663,710	3,757,419	5,259,140	5,309,030	4,771,050	4,206,085
Isle Royale	*Nil.*	2,171,955	3,569,748	3,134,601	2,442,905	2,973,761
Mass	*Nil.*	837 277	2,345,805	2,576,447	2,182,931	2,007,950
Michigan	....	....	166,898	275,708	2,746,127	2,891,796
Mohawk	....	160,897	226,824	6,284,327	8,149,515	9,387,614
Osceola	11,200,000	13,723,571	13,416,398	16,059,636	20,472,439	18,938,965
Phoenix	....	93,643	....	202,823	1,162,201	273,219
Quincy	14,116,551	20,540,740	18,988,491	18,498,288	18,343,160	18,827,557
Tamarack	18,400,000	18,000,852	15,961,528	15,286,093	14,961,885	15,824,008
Trimountain	....	....	5,730,807	9,237,051	10,211,230	10,476,462
Winona	....	....	101,188	1,036,944	646,025	*Nil.*
Wolverine	4,778,829	4,946,126	6,473,181	8,999,318	9,764,455	9,464,418
Others	4,000,000	640,591	700,067	75,000	50,000	....
Totals	144,227,346	155,507,465	170,194,996	192,299,485	208,392,485	218,999,753

county, but it is uncertain whether the openings will become extensive enough to figure in the production this year.

Osceola's decreased yield in 1905 was due to labor strikes and the explosion of a powder magazine underground in the North Kearsarge branch, which caused a suspension of work there for some time. The Alloucz was the only new producer in 1905. It started shipping rock to the Centennial stamp mill in the summer. While its tonnage was not large, the richness of its workings enabled it to make a favorable showing. The Phœnix is no longer a producer, the mine having been closed down. Winona discontinued regular production, but secured some copper from mass and barrel work. Atlantic's yield was the smallest for many years; this was due to a change in the system of underground operations.

It is safe to say that the output in 1906 will be fully as large as in 1905. Several of the newer mines will turn out more. The Champion, Trimountain and Baltic are certain to increase their production under normal conditions. The Mohawk is compounding its stamp and will make a gain of 50 per cent. as soon as its mill is remodeled throughout Centennial and

Ahmeek, two Kearsarge lode properties, are just in their infancy, and their production will grow as development work advances and openings underground are extended. Among the old mines there is also promise of increased yield. The Calumet & Hecla company's activity in opening the Kearsarge and Osceola amygdaloid beds where they traverse its main property at Calumet will in time furnish an important production. With the completion of the equipment at No. 8 shaft on the Mesnard property of the Quincy Mining Company much larger rock shipments will be possible. When the new system of mining is in use throughout the Atlantic, its production will be larger than ever before. The Osceola should make a better showing in 1906. The milling capacity has been enlarged by changing the heads from the simple to the compound type.

Dividends declared in 1904 and 1905 were as follows :

Mine.	1904.	1905.	Mine.	1904.	1905.
Atlantic	$	$ 50,000	Osceola	$192,300	$384,600
Baltic	......	1,250,000	Quincy	500,000	600,000
Calumet & Hecla	4,000,000	5,000,000	Tamarack	90,000	120,000
Central	......	160,000	Wolverine	450,000	660,000
Champion	200,000	1,000,000			
			Total	$5,432,300	$9,224,606

The Baltic was the only new dividend-payer in 1905. The Central's disbursement represented the payment to its stockholders in liquidation, that amount having been received from the Calumet & Hecla for the property. In addition to the companies given in the above table, there were dividends of $300,000 by the St. Mary's Mineral Land Company and $1,540,000 by the Copper Range Consolidated Company. The Copper Range Company paid no dividends in 1904, while the St. Mary's paid $150,000 in that year. As both of these companies secure their profits from holdings in mines whose payments are given in the foregoing table, their payments are not included in the totals for the district.

In the following table the dividend disbursements are given for every half decade from 1850 to 1900, and all years succeeding:

Year.	Dividends.	Year.	Dividends.	Year.	Dividends.	Year.	Dividends.
1850	$ 84,000	1870	$ 700,000	1890	$3,415,000	1902	$3,440,000
1855	168,000	1875	1,920,000	1895	3,280,000	1903	4,980,000
1860	120,000	1880	3,080,000	1900	9,811,200	1904	5,432,300
1865	510,000	1885	1,970,000	1901	7,496,900	1905	9,224,600

Total dividend payments by Lake Superior copper mining companies since the first disbursement, made by the Cliff, in 1849, have been as follows, the figures being to the end of 1905: Atlantic, $990,000; Baltic, $1,250,000; Calumet & Hecla, $92,350,000; Central[1], $2,350,000; Champion[2], $1,500,000; Cliff[3], $2,518,620; Copper Falls[1], $100,000; Franklin, $1,240,000; Kearsarge[4], $160,000; Minnesota[5], $820,000; National[1], $320,000; Osceola, $4,824,000; Pewabic[6], $1,000,000; Phœnix[1], $20,000; Quincy, $15,220,000; Ridge[7], $100,000; Tamarack, $8,700,000; Trimountain[8], $300,000; Wolverine, $2,430,000; total, $136,972,820.

[1]Idle. [2]Owned equally by Copper Range and St. Mary's. [3]Absorbed by Tamarack. [4]Absorbed by Osceola. [5]Absorbed by Michigan. [6]Absorbed by Quincy. [7]Absorbed by Mass. [8]Absorbed by Copper Range.

There are upwards of 18,000 men employed in the mines, stamp-mills and smelters of the Lake Superior copper district at the present time; never has the force been so large. Working forces of the mines for the last 10 years are given in the following table. These figures, however, do not include the many thousands of men employed on the railroads; by Lake steamers during the season of navigation; in the iron foundries and machine shops; in the forests, saw-mills, etc., all of whom are dependent upon the mines:

Year.	Number.	Year.	Number.	Year.	Number.	Year.	Number.	Year.	Number.
1896	8,170	1898	10,469	1900	13,971	1902	15,130	1904	16,000
1897......	8,726	1899	13,051	1901	14,498	1903	15,150	1905	18,000

Some idea of the widespread interest in Lake Superior copper mining companies may be gained from the list of stockholders. At the present time there are 23,000 shareholders in the active companies organized under the Michigan laws, besides the stockholders in companies organized under the laws of other States. In 1896 there were only 6598. There are nearly 4000 stockholders of the Calumet & Hecla company alone.

Beginning at the extreme northeast, in Keweenaw county, and ending in Ontonagon county, the present status of the mines, briefly stated, is as follows:

At the Ashbed a few men are employed sinking below the old tunnel level. Some copper has been exposed. The consolidation of this property with the Arnold and Meadow, which are under the same management, is talked of, and undoubtedly would be to advantage.

The Keweenaw Copper Company, which was formed this year by Charles A. Wright, of Hancock, who was the original promoter of the Copper Range enterprise, is doing diamond drilling on one of its properties, the Mandan-Medora. The Montreal River lode has been located, and the ore showed strong mineralization. The Keweenaw company has built the Keweenaw Central Railway from Lac La Belle, and will push it on to Calumet in 1906. Thomas F. Cole, president of the Oliver Mining Company, and John D. Ryan, managing director of the Amalgamated Copper Company, and associates, have purchased controlling interest in the Keweenaw Copper Company, and its subsidiary concern, the Keweenaw Central Railway. Important developments are likely to result from the aggressive work which is projected for 1906.

Resolute, comprising 1120 acres and located between the Mandan-Medora and the Delaware, has been purchased by the Cole-Ryan interests. Development work will likely be started in the summer of 1906. The Kearsarge and Montreal River beds traverse the property.

At the Delaware, now owned by the Calumet & Hecla, an old pit on the Montreal River lode was cleaned out and converted into a shaft. Sinking has progressed to a depth of 150 ft., with a remarkably good showing of

copper. Diamond drilling to determine the strike and incline of the lode is under way.

Phœnix was abandoned in the middle of 1905, as the St. Clair vein became too narrow.

Diamond drilling on the Cliff, owned by the Tamarack, located the Kearsarge amygdaloid.

The Miskwabik discontinued sinking and drifting, and rigged up a diamond drill to determine more fully the identity of the lode upon which work has been done. Drilling was carried to a depth of 500 ft., but revealed a poor copper showing.

Mohawk is now making good returns, and the outlook for 1906 is bright. The installation of the fourth head at the stamp-mill completed the latter. The new head is of the steeple-compound type and it has been decided to change the other stamps to this form. When completed, the change will permit an annual production of upwards of 18,000,000 lb. copper yearly. The Mohawk declared its first dividend, $2 a share.

Ahmeek is more than paying expenses, besides developing its mine in a thorough manner. Its rock is being shipped to the Tamarack stamp-mill and averages considerably better than the Osceola's North Kearsarge mine, next south. Permanent equipment is being installed and the Ahmeek eventually will become one of the good mines of the district. Present production is from rock taken out in opening the shaft and drift stopes. The mine will make an increased production in 1906.

Allouez has been put on a paying basis. Since opening the Kearsarge lode at a depth of 1500 ft., five levels have been opened. Production started in the middle of 1905, rock being shipped to the Centennial mill. Allouez's No. 2 shaft will be sunk as rapidly as possible, and should be bottomed in the lode and on a producing basis in two years.

Wolverine makes the lowest-cost copper on Lake Superior. Its production will continue about 10,000,000 lb. per annum for 20 years or more, as no increase is contemplated.

Centennial's energies are being devoted mainly to the opening of new ground. No. 1 shaft is bottomed in what is believed to be the continuation of the upper limits of the rich copper-bearing shoot which makes southward from the Wolverine on the Kearsarge lode.

Small forces are employed at the Old Colony and Mayflower properties, eastward, exploring for copper-bearing lodes. Work at both properties is at considerable depth, and results at one are of importance to the other, as they carry many of the same lodes.

At the Calumet & Hecla much energy is being devoted to the development of virgin ground. Gradually work is being resumed in all the Osceola lode shafts, Nos. 13, 14 and 16 now operating. The remodeling of the stamp-mills progresses and will be completed in 1907. The portion of the

mills under reconstruction are being fitted to treat amygdaloid rock, which indicates that it is planned to carry the Osceola and Kearsarge lode work forward permanently. The shaft started on the extreme southeastern portion of the property has struck the Kearsarge lode with encouraging results. Permanent equipment is being installed. The two Kearsarge shafts farther north are being opened steadily. Provisions for substituting electricity for steam as an operative power are progressing steadily, and will be completed in 1906. A great saving should result, as there are many small engines, pumps, etc., and in piping the steam much power is lost. The Calumet & Hecla has acquired extensive holdings of promising mineral lands in Keweenaw county, and organized two subsidiary corporations to develop them. An interest in the Superior has been purchased.

Diamond drilling located the Kearsarge lode on the Laurium lands, but nothing further resulted. Tecumseh became active in 1905. Two shafts have been opened on the Kearsarge amygdaloid, which was located with the diamond drill, and the outlook is promising.

Rhode Island started diamond drilling for the Kearsarge amygdaloid bed; the situation offers encouragement. Work on the Pewabic amygdaloid continues through No. 2 shaft, the lode showing considerable copper in spots.

Franklin cut the Kearsarge amygdaloid with diamond drills. A crosscut will be extended from the workings on the conglomerate to test the formation. One of the most important developments in recent years by the Franklin is the striking of rich ground in the bottom of No. 1, the rock being better than anything heretofore opened. The old Franklin property is still active.

Arcadian has started a shaft upon a lode located with the diamond drill a few years ago. Much of the extensive surface plant was sold, but there is enough machinery left to suffice for the temporary work that is now under way.

Quincy's outlook is improved by the approach of No. 8 shaft to the producing stage. This shaft is on the Mesnard branch, being separated from the old mine on surface by the Franklin, but connections will be established between the two properties under the Franklin workings.

Isle Royale started work on the Baltic amygdaloid. A shaft is being sunk upon the outcrop. The shaft on section 11, which is developing the Isle Royale lode, is opening favorable ground and furnishes considerable rock for the mill. Operations in the active shaft at the north end of the mine continue and some exploratory work has been done on the Grand Portage lode.

Superior, on the Baltic lode in section 15, passed into the hands of the Calumet & Hecla.

Atlantic is developing the Baltic lode on its section 16 tract. A shaft was

started from surface and a drift is being extended on the vein from the Baltic workings at the sixth level. The outlook is good. Work of changing the system of mining in the old mine progresses. Lower costs will result.

Baltic is equipping and developing its No. 2 shaft, which should enter the producing list in the summer of 1906. At the stamp-mill the simple stamps have been changed to steeple-compound.

Trimountain is developing new ground. Its heads at the stamp-mill are being compounded and the milling capacity thereby increased. Champion's development work is progressing steadily. The mine shows no reduction of value as depth increases.

A portion of the Globe, immediately south of the Champion, is under option to the Copper Range company. The shaft should encounter the lode 250 ft. from surface. Openings south from the Champion indicate that the Globe contains good copper ground.

Drifting on what is believed to be the Baltic lode is being pushed by the St. Mary's Mineral Land Company at its Challenge mine. Some copper is exposed and the situation is not without promise.

The Erie-Ontario Development Company, a new concern, is exploring for the Baltic lode at a point about half-way between the Challenge and Winona mines.

Elm River is sinking its No. 1 shaft. Wyandot is exploring, but nothing of value has yet been developed.

Winona's openings, particularly in the lowest level of No. 3 shaft, are improving. The management is confining its efforts to a thorough exploration of the property.

South of Winona, the St. Mary's Mineral Land Company and other property owners have united in organizing the King Phillip Mining Company to develop lands traversed by the Winona lode.

No. 4 shaft, started by the Adventure the latter part of last year, is promising and there are still hopes of making a paying mine.

Michigan (Ontonagon county) is making a good showing on the Branch vein, and a fair-sized product is turned out between the rock stamped at the Mass mill and the yield of heavy copper. The Mass Consolidated has extended its railroad to C shaft, which will become a producer shortly. The old Belt property has been taken over by the Lake Copper Company, and will be explored. Victoria's stamp-mill and other surface work is rapidly nearing the time when the property will be enabled to start producing. The Copper Crown Mining Company is exploring the Norwich by means of a tunnel driven into the base of a bluff on the property.

Adventure produced 2,700,354 ℔. of mineral containing 1,606,208 ℔. of fine copper in 1905, which was sold (at average of 15.72c.) for $252,572. Mining, taxes, smelting and construction, including the new shaft, cost

$341,223 (21.24c. per lb.), showing a loss on the year's work of $88,651.

Osceola in 1905 mined 1,128,376 tons of ore, and milled 1,007,200, which yielded 24,854,391 lb. of mineral and 18,938,965 lb. of refined copper, the latter selling for $2,942,239 (15.54c. per lb.). The cost of production was $2,022,625 (10.68c. per lb.). Mining cost $1.39 per ton of ore milled; milling $0.17. The corresponding figures for 1904 were $1.32 and $0.16.

The Osceola mine continues to show an improved grade of ore. The diminution in output is explained by loss of time during which workmen were on strike.

The South Kearsarge ore also continues of good grade. North Kearsarge is losing its former richness; the "disturbed area" continues to be exposed on all the new levels and a large amount of exploration is necessary to find its limits. During the year the mill was improved by the replacing of simple by compound stamps, and the addition of regrinding and concentrating machinery. The present equipment consists of six compound stamps, each assisted by a set of 16x36-in. rolls and their concentrating apparatus, all ore being crushed to pass a $^3/_{16}$-in. round hole, instead of a $\frac{1}{4}$-in. hole, as before. These improvements have increased the mill's capacity by 25,000 tons per month and have reduced the cost per ton for stamping.

The Quincy mined 1,222,257 tons of ore in 1905, hoisted 1,168,519 tons and milled 1,135,162 tons, which yielded 26,506,368 lb. of mineral and 18,827,557 lb. of refined copper. The copper sold for $2,981,121 (15.83c. per lb.) and the cost of production was $2,044,656 (10.86c. per lb.). Dividends amounted to $600,000. The cost of mining and milling per ton of rock hoisted was $1.47, of which the cost of milling was probably about 20c. (not stated in the official report).

Electric locomotives are now used to haul ore from No. 4 to No. 2 shaft, where it is hoisted, and from No. 8 to No. 6 shaft. The 49th level is to be equipped throughout with electric traction.

Stamp mills Nos. 1 and 2 treated 378 tons of ore per day more than they did during 1904. One head at No. 2 mill was reinforced by a pair of Nordberg rolls to crush the oversize, and its output was thereby increased by 100 tons per day. The Nordberg steeple compound stamp has not fulfilled expectations as to efficiency.

Missouri.—H. Foster Bain and E. O. Ulrich, in *Bulletin* No. 267 of the U. S. Geological Survey, stated that copper is now mined and smelted near Sullivan, Mo., and copper deposits at other points in the State are attracting attention.

Attempts to mine sulphide and carbonate copper ore have been made in Missouri since 1837, and at different times copper furnaces have been operated in Shannon, Ste. Genevieve, and Crawford counties.

With the exception of one or two sporadic occurrences of merely miner-

alogic interest, copper is found only in the southern part of Missouri, within the region broadly known as the Ozark uplift.

The ores show a preference for certain stratigraphic horizons, and, being bedded, may be prospected with ease and economy. The common association of sulphides with specular iron of the sandstone region points to the advisability of the investigation of the old iron pits. While no large copper deposits will probably be found, the low cost of flux, fuel and labor makes it possible to work some, at least, of the deposits with profit. In Shannon county the most favorable localities are along the contact of porphyry and dolomite at points where the conglomerate beds at the base of the latter fill in shallow basins in the crystalline rocks. The mines near Ste. Genevieve have not been worked for several years, but the building of the Illinois Southern railway to within a few miles of them so changes the situation that they can probably now be worked with profit.

Montana.—As heretofore, the copper production of this State continued to be derived chiefly from Butte. The latter camp has apparently received a new lease of life, the extent of the ore-bearing formation having been proved by new discoveries to be much greater than was formerly the opinion, while at the end of the year the opening of the downward extension of the Anaconda vein, by a cross-cut on the 2200 ft. level, showing a thick body of rich ore, established great confidence in the future of the district with increased depth. During 1905 the old Speculator mine, now owned by the North Butte company, proved to be a bonanza and made a large output, the ore being smelted at the Washoe works of the Amalgamated.

During the latter part of the year negotiations were in progress between F. A. Heinze and Thomas F. Cole, and in the early part of 1906 these were consummated by the transfer of the property of the United Copper Company (Heinze) to Cole, who organized the Butte Coalition Mining Company to take it over. The consideration to the United Copper Company is understood to have been $25,000,000, one half in cash and one half in securities. Cole and his associates being affiliated with the Amalgamated interests, the litigation between the previously conflicting interests was terminated and the various property rights were adjusted on a side-line basis.

In metallurgy, the year was signalled by the continued successful operation of the huge reverberatory furnaces at the Washoe works, the successful introduction of blast furnaces upward of 50 ft. in length at the same works[1], and the putting into operation of the Baggaley process at the works of the Pittsburgh & Montana company, which appears for the first time as a producer of copper.

Montana as a State and Butte as a district continue to hold the premier

[1]These are described in a subsequent section of this article.

positions among American copper producers. As a single district Butte is far in the lead of any other.

The Raven and Reins produced some ore, but not enough to increase the total much. Both are expected soon to increase their outputs, especially Raven, which is driving a cross-cut from the 1500-ft. station of the Buffalo to cut its three veins in Snoozer ground. Reins spent the larger part of the year in development work and will not finish what it has on hand until the middle of 1906. Pittsburgh & Montana did not become a producer until near the end of 1905.

At the end of 1905, the Amalgamated was mining more ore than ever before. Its daily shipments to the Washoe plant aggregate 7800 tons, of which 7500 come from its own mines and 300 from small properties in the district. Boston & Montana is sending 112,000 tons a month to Great Falls.

Considerable sinking was done during 1905; a total of 200 ft. was added to the shaft of the Anaconda; 406 to the Neversweat; 122 to the St. Lawrence; 200 to the West Steward; 200 to the Original and 200 to the West Colusa. The Anaconda cross-cut its main vein on the 2200 ft. level and is now driving for it 200 ft. deeper.

Boston & Montana began extensive improvements early in 1905 and will complete them in 1906. It is sinking a new 1200-ft. shaft on the Leonard and will equip it with a new Nordberg hoisting engine and steel gallows-frame. The work has not interfered with the operation of the mines, the ore of which is raised through the Leonard. The Mountain View was equipped with a new engine and the boiler capacity increased.

The United Copper Company mined between 1100 and 1200 tons of ore per day. Its producers were the Rarus, Cora, Minnie Healey and Belmont mines. The company takes all the ore it can get in the open market.

W. A. Clark mined and treated an average of 900 tons of ore a day in 1905, all coming from the Steward and Original mines, both of which are 2000 ft. deep and contain good veins of medium and high-grade ore. Each shaft was sunk 200 ft. Extensive improvements were begun at the reduction plant. Two stands of 85x126-in. converters are to take the place of the old blister copper furnaces, and Chilean mills and Overstrom tables for reworking the tailings have been added to the dressing works. The concrete-steel stack, 352 ft. high, was finished in November.

The North Butte company came into existence last spring, its promoters buying the Speculator mine for $5,000,000. Since then it has increased its possessions materially, re-timbered the old shaft from top to bottom, 1600 ft., and ordered a new hoisting engine and steel gallows-frame. While the shaft was undergoing repairs the output of the property was about 550 tons a day, but now it is 700 to 800 tons. The Berlin group of

claims has been purchased. North Butte will explore the ground from its deep workings.

The Pittsburgh & Montana became a producer in the fall and bids fair to increase its output. The smelter at present has only one blast furnace and one converter in operation. Toward the latter part of the year several new companies were organized to operate in Butte. The East Butte was first to enter the field. Others have been the American Consolidated Copper Company (Coram syndicate), Butte & London Exploration Company and Butte Copper Exploration Company. The last intends to work the Six O'Clock group of eight claims on the east side of the district. There is a shaft 440 ft. deep on the Six O'Clock.

Nevada.—One of the most important developments in copper mining in 1906 was the recognition of the great value of the deposits of low-grade ore at Ely, which are now held to be sure to produce a large amount of copper in the near future. This is undoubtedly one of the most promising camps of the Rocky Mountains, and is likely to take rank with Butte, Bingham, Bisbee, and Globe as one of the great copper camps of the United States, while its mines will be rated among the great ore deposits of the world.

The principal developments have been made in the mines owned by the Nevada Consolidated Copper Company, the stock of which was placed on the Boston and New York markets in the early part of 1905. Other mines of the district are those of the Cumberland-Ely and the Giroux Consolidated companies, neither of which, however, approaches the magnitude of the Nevada Consolidated so far as yet known. The Cumberland-Ely company is controlled by the Guggenheim interests, and through that company control of the Nevada Consolidated was secured in the early part of 1906. The conditions at Ely are outlined in the following report by J. Parke Channing, on the property of the Nevada Consolidated, this report being dated Aug. 3, 1905.

The properties of the Nevada Consolidated Copper Company are located in the Robinson district, about six miles west of Ely, White Pine county. Approximately this is 150 miles south of Toano, Nevada, on the Southern Pacific Railroad. At present there is no railway connection. Surveys have been made and it is proposed to build a standard gage road from Toano to the mines.

I am of the opinion that with an expenditure of $1,225,000 to build and equip the railway from Toano to the mines and with a further expenditure of $975,000 to equip the Ruth mine, build a concentrator, a powerhouse, a smelter and the necessary shops, buildings, etc., making a total of $2,200,000, the property can be put in condition to mine and treat 1000 tons of ore per day, which would produce about 13,700,000 ℔. of copper per annum.

By the further expenditure of $500,000 for the equipment of the Eureka mine and the enlargement of the mill and smelter, another thousand tons of ore per day could be treated, which would yield about 12,000,000 ℔. of copper additional per annum.

There is developed in the two mines between 5,000,000 and 6,000,000 tons of ore. The conditions in the district, however, all point to the probability of the extension of the orebodies already opened up, and of the finding of new orebodies. I, therefore, am of the opinion that one may safely figure on a daily production of 2000 tons of ore for many years to come, and there is a strong possibility that this production might even be increased to 2500 or 3000 tons per day without materially affecting the ore reserves.

However, on an assumption of only 2000 tons of ore per day and calling the annual production 25,700,000 ℔., cost of production, allowing for construction, will be as follows: Total cost per pound copper sold in New York, 7.53c.; or 7.03c., deducting 50c. for gold and silver recovered.

The mining lands of the company may be divided into two groups. The first is called the Ruth, and contains approximately 300 acres. The second is called the Eureka; it lies about one mile west of the Ruth and contains approximately 400 acres.

During the summer of 1904, a careful geological survey of the district was made by Prof. Andrew C. Lawson, of the University of California. His report shows that the copper-bearing rock of the district is an immense porphyry intrusion, cutting through the limestones and shales of the country. This porphyry is a white silicious rock which on the surface is weathered brown and shows but little trace of copper. Where orebodies have been developed, as a general rule the first 50 ft. to 100 ft. of the porphyry has been leached and carries not to exceed 0.5 per cent. copper. On getting below this zone of oxidation, the ore becomes white and comparatively soft in texture, so that it is drilled and blasted with great ease. The principal minerals contained in the silicious gangue are pyrite and chalcocite. These minerals occur in fine seams and veinlets and also in small masses with which the ore is peppered.

The walls of the orebodies are not well defined, pay-ore gradually merging off into that which is too low to be commercially valuable; it is doubtful even if the long cross-cuts in either the Ruth or Eureka mines have passed beyond the limits of pay-ore. At the Ruth mine the orebody varies from 50 ft. to 250 ft. in width, and developments have so far shown from 400 ft. to 900 ft. in length, and a vertical height of 250 ft. At the Eureka mine the orebody has been opened up for a length of 700 ft. east and west, and 800 ft. north and south, and has been proven to be of commercial grade for a depth of at least 100 ft.

Not one-hundredth part of the porphyry exposed on the claims of the

company has been explored, and therefore it is difficult to say how much of this porphyry may not be mineralized. The 50 ft. to 100 ft. of surface oxidation has deterred prospecting, and it is for this reason that the enormous ore deposits of the district have been undiscovered until recently. The geologic conditions and the ore deposits are very similar to, in fact almost identical with, those in the Clifton district of Arizona. The ore-bodies will probably be found of great lateral extent, their upper boundary will be from 50 ft. to 100 ft. below the surface and the lower boundary will be from 300 ft. to 400 ft. below the surface, thus giving an average thickness of ore from 250 ft. to 300 ft. vertically. This same phenomenon of great lateral extension and comparatively small depth also exists in the Clifton district.

The Ruth ore will average 2.6 per cent. copper; 0.02 oz. gold and 0.05 oz. silver per ton. The Eureka ore will average 2.2 per cent. copper; 0.02 oz. gold and 0.03 oz. silver per ton.

The Ruth mine undoubtedly will be worked on the top-slice caving system, which is now employed in many of the Lake Superior hematite mines. The physical character of the ore and its method of occurrence under-ground makes it peculiarly fitted for this method of mining, in which the cost of mining and the loss of ore are reduced to a minimum. The topography at the Eureka mine is such that the whole of the orebody as at present developed can be stripped with a steam shovel, and the ore either mined by the same method or by what is known as the milling system. For purposes of calculation I have assumed the milling system, which consists in undercutting the ore, say at a depth of 100 ft., with drifts and cross-cuts after it has been stripped, and then putting up raises through to the top. The ore is then simply blasted down these raises, trammed out in cars to the shaft, hoisted to the surface and there sent to the mill in the regular railway cars. This method of mining was used very successfully on two of the Mesabi Range mines.

From the mines the ore will be brought by standard gage cars to the concentrating mill, which will be located just north of Ely. Here a sloping site exists where the water of Murry creek may be brought in by gravity, and where unlimited dumping room for the tailing is available.

During the summer of 1904, an experimental mill was erected at the Ruth mine, and it is from the results of the runs in this mill that the new 1000-ton mill has been designed. The Ruth ore will yield 77 per cent. of its copper; the Eureka, 79 per cent. The experiments indicate that the Ruth ore will concentrate about 7:1, with concentrates running 13.3 per cent. copper, and the Eureka ore about 10:1, with concentrates running 17.5 per cent. copper. The average composition of the concentrates will be approximately as follows: Silica, 28 per cent.; copper, 15 per cent.; iron, 24

per cent.; alumina, 4 per cent.; sulphur, 26 per cent.; balance, 3 per cent.; total 100 per cent.

The smelting plant will comprise McDougall roasters and reverberatory furnaces, where they will be smelted to a 50 per cent. matte. The reverberatory furnaces will be provided with waste heat boilers. The slag will be granulated and the matte will be converted to blister copper. The total extraction per ton of Ruth ore should amount to 37.5 lb. or 72 per cent.; of Eureka ore, 33 lb. or 75 per cent.

[The mill, smelter and railway of this enterprise are now under construction, and will be completed during the present year.]

New Mexico.—No important copper producing district has yet been developed in this Territory, but there is a production from many small mines, which in the aggregate is considerable. The mines of the Santa Fé Gold and Copper Company, at San Pedro, Santa Fé county, continued in idleness in spite of the high price for copper, it being apparently recognized that these mines cannot be profitably operated under any ordinary conditions. A small amount of copper occurs in the zinc ore of the Magdalena, or Kelly, district, some of which is shipped to Missouri and Kansas for the manufacture of zinc oxide, the copper remaining in the cinder, which is sold to other smelters. In the fall a large vein of ore high in copper (cuprite and native) was opened in the Graphic mine and 40 to 50 car loads were shipped. It is expected that a considerable production will be made from this vein in 1906.

(By Charles R. Keyes.)—Western and southwestern New Mexico is rapidly attaining importance as a copper producer. A number of significant developments have been made. As yet, the most important copper districts are at considerable distances from railway facilities.

Lines of railroad have been located here during the past year; these tap a number of these camps. The Santa Rita mines continue to be good producers. In the Mogollon region, bornite ores (which occur in large deposits) are finding their way to the markets in spite of transportation drawbacks.

The low-grade copper ores of the great "Red Bed" belt have received attention. This formation covers large areas in New Mexico; nearly everywhere it carries more or less copper. In a small way, copper has already been extracted from the Red Beds. Large projected developments are being regarded with much interest.

North Carolina.—The copper production of this State in 1905 was small. The Union mines at Gold Hill, Rowan county, were the principal producers. Experiments with the Garretson smelting furnace were made at the Ore Knob mine, near New River, Ashe county, during the year, but apparently they were not carried to a conclusion. There was no significant production of ore in the Virgilina district, on the boundary line between North Carolina and Virginia. A small amount of southern

ore was smelted at the works of the Eustis Mining Company, near Norfolk, whither cinder from copper-bearing pyrites, originating at Capelton, Canada, is shipped for treatment after previous burning for sulphuric acid manufacture at works in Massachusetts.

The Union Copper Mining Company, of Gold Hill, began shipments of ore in March and during the summer attained an output of about 1000 tons per month. This was chiefly obtained in development work. The ore averages about 4 per cent. copper, which occurs as chalcopyrite in a quartzose gangue. Up to July the output was shipped to the American Smelting and Refining Company at Perth Amboy; since that time most of it has gone to the Tennessee Copper Company at Isabella, Tenn., where it is valuable as a silicious flux for the pyrrhotite.

The vein, which is 20 to 40 ft. wide in some places, has been opened by eight or ten shafts to a depth of 150 to 650 ft., along a distance of nearly a mile on the surface. The ore occurs in lenses or chimneys. Two oreshoots that are being worked at present are each about 100 ft. long. With a production of about 2500 tons a month, it is hoped to be able to mine for about $1.70 a ton. It is reckoned to be cheaper to ship the ore crude rather than to attempt concentration. A saving of over 90 per cent. copper would have to be made in concentrating, in order to be as profitable as crude-ore shipments.

Oregon.—A small production of copper has been made in this State through the Oregon Smelting and Refining Company, which has a 150-ton smelter at Sumpter, Baker county. This plant was in operation in 1905, doing exclusively a custom business in ores from adjacent territory. Some ore was also obtained from the Seven Devils district, Idaho, especially in the latter portion of the year.

Tennessee.—The Ducktown district continued to be the only producing center of this State, and as heretofore the production was by two companies, these being the Tennessee Copper Company and the Ducktown Sulphur, Copper and Iron Company. The statistics of production for this State have been given in the table in the first part of this article. The record of the Tennessee Copper Company is given in the following abstract of the report of its directors and officers for 1905. This company publishes an annual report, which in technical value exceeds that of most companies, and is especially valuable in view of the important technical progress that is being made at its works, under its accomplished president, J. Parke Channing, and his able assistants.

The Tennessee Copper Company has altogether abandoned heap roasting, and now smelts all its ore pyritically. During 1905 it smelted 229,116 tons of fresh ore (total charge 368,353 tons) with 19,473 tons of coke and recovered 7,977,982 lb. of copper in bessemer pig and shot. None of the product was refined. The cost of producing the copper, f. o. b. cars, was

8c. per lb., or \$2.8127 per ton of ore; the corresponding figures in 1904 were 9.18c. and \$3.27. Subsequent expenses brought the cost up to 9.28c. The cost of smelting in 1905 was \$1.4763 per ton.

Utah.—The copper production of this State increased materially in 1905, and there is prospect for a much larger increase in 1906. Practically the whole output was derived from the Bingham district, which is now recognized as one of the great copper producers of the United States, but a portion came from the Cactus mine, belonging to the Newhouse interests. The producers in the Bingham district are the following, those having smelting plants being marked by a single asterisk, and those having converting plants being marked with two asterisks: Utah Consolidated (Highland Boy)**, United States Mining, Smelting and Refining Company**, Bingham Consolidated**, Utah Copper Company, Ohio Copper Company, Yampa*, and Boston Consolidated. The American Smelting and Refining Company has copper smelting furnaces, producing matte from ore from the other companies, and is now erecting a large smelting plant at Garfield on Great Salt Lake. The Highland Boy and Yampa smelters have reverberatory furnaces. The Bingham and United States have blast furnaces, but the latter is also installing reverberatories. The new Garfield plant will have both kinds, but chiefly reverberatories. It is generally recognized that the latter are ahead in the smelting of the Bingham ores, which contain a large proportion of fines. This proportion will be increased by the concentrates from the new mills.

The Bingham ores occur in two forms, namely, in large masses in metamorphosed limestone and in grains disseminated through monzonitic intrusives. The orebodies of the former class occur in the form of lenticular beds lying roughly parallel with the stratification of the country rock. These beds are localized into elongated lenticular shoots, which dip roughly with the bedding and pitch moderately. These shoots are sometimes of great size, being several hundred feet in length along their strike, nearly 200 ft. thick, and have been followed downward continuously for several hundred feet. The ore is essentially pyrite, containing from 2 to 4.5 per cent. copper, 2 to 5 oz. silver and 10c. to \$1 gold. Associated with the copper deposits there are bodies of silver-bearing galena, containing more or less zinc blende.

The disseminated copper ore occurs throughout extensive stocks of monzonite, but particularly in areas where it is fractured, fissured and altered. In such zones there are found irregular grains of copper-bearing pyrite in small veinlets, chiefly along joint or fracture planes. Definite shoots of this mineralization have not been proved.

The large copper production of Bingham has heretofore been derived chiefly from the lenticular deposits first mentioned, such being worked by

the Utah Consolidated Mining Company (Highland Boy mine), the United States Mining Company (Old Jordan, Telegraph and Galena mines), the Bingham Copper and Gold Mining Company (Commercial, Dalton and Lark mines) and the Boston Consolidated Mining Company. More recently the production of copper from the disseminated ore in the monzonite has been inaugurated by the Utah Copper Company, which erected a 500-ton dressing works in main Bingham cañon. The operation of this property has been successful, and the erection of a new mill of 6000 tons daily capacity on the shore of Great Salt Lake is now in progress. The success of the Utah Copper Company led the Boston Consolidated Mining Company to undertake the exploitation of the large bodies of cupriferous monzonite which exist in its property, and a mill of 5000 tons daily capacity is to be erected by it on the shore of Utah lake in 1906. This will consist of two units of 2500 tons each.

This monzonite of Bingham contains from 1 to 1.4 per cent. copper. In a statement by Samuel Newhouse, president of the Boston Consolidated, to his stockholders, advising the erection of a 2500-ton dressing works (which has now been decided upon), the cost of production is estimated as follows: Mining with steam shovel, 40c. per ton; milling, 35c.; smelting, 20c.; freight and refining, at 2c. per ℔. of copper, 42c.; selling and general expense, 16c.; total, $1.78. Estimating a copper content of 1.4 per cent. in the crude ore and a recovery of 75 per cent. in dressing, the yield per ton of ore would be 21 ℔. of copper, worth $2.52, with copper at 12c. per ℔. and 20c. for gold and silver value, making a total of $2.72, giving a net value of $0.94 per ton of crude ore. The cost of plant of 2500 tons daily capacity is estimated as follows: Railway, $350,000; railway equipment, $100,000; power plant for 5000 tons, $150,000; mill, $450,000; two steam shovels, $25,000; reservoirs, etc., $50,000; total, $1,125,000, on which interest at 6 per cent. and amortization at 10 per cent. would come to about 24c. per ton of crude ore.

The working of the very low-grade copper deposits of Bingham will introduce there a condition of mining analogous to what exists at Lake Superior, where immense tonnages of copper ore of very low grade are worked, the Utah mines having the great advantage, however, that their ore can be extracted to a large extent by open-cut work. Other companies in the district, besides the Utah Copper Company and the Boston Consolidated, have large deposits of this low-grade ore.

It is now recognized that the disseminated ores of Bingham are the most important of the deposits of that district. Except in the case of the Highland Boy, the heavy sulphides have not been highly profitable; in some cases, e. g. the United States ores, the margin is very close, their chief value being as a basis for the smelting of silicious ore from other districts. The tonnage of the disseminated ores is enormous. They are of very low

grade and require fine grinding (30-mesh), but they can be mined cheaply and yield a fairly high proportion of their mineral by ordinary concentration. The control of the Utah Copper Company was purchased in 1905 by the American Smelters' Securities Company. Negotiations were inaugurated for a consolidation with the Newhouse interests (Boston Consolidated), but these appear to have fallen through. The concentrates from the new dressing works will constitute a large part of the ore supply for the Garfield smelter, which will have an initial capacity for 1500 tons per day and will go into operation in the summer of 1906.

The developments in the Bingham district in 1905 were of such magnitude and so rapid that existing smelting capacity was exceeded. Indeed, the smelters found it necessary to ask producers to curtail their output temporarily. Incidentally, a disastrous fire at the Pleasant Valley Coal Company's mines at Sunnyside, Utah, cut off the supply of coke and for several weeks the plants in the Salt Lake valley came dangerously near a complete shut-down.

The Boston Consolidated began in March to develop the disseminated ore within its territory. Thirteen tunnels were driven, and nearly two miles of underground workings, in addition to a lot of prospecting on the surface, with no evidence of a bottom to the ore. Enough is said to have been done in three of these tunnels—the Ben Hur No. 1 and No. 2 and the Metropolitan—to justify the statement that, in the immediate zone penetrated by these adits, over 30,000,000 tons of ore is available for extraction. It is the intention of the company to inaugurate mining by steam shovels during the early summer of 1906. Three shovels have been ordered for this work. It is expected to begin construction of the mill during the spring.

The Utah Copper Company management claims that in its mines there is now available more than 35,000,000 tons of porphyry ore. The following is an abstract of the official report of the company for the year ending June 30, 1905.

The orebody within the Utah Copper Company's properties is a mass of copper-bearing, altered porphyry, the dimensions of which, either with respect to the vertical or horizontal extensions, or its exact shape, have not yet been determined. Extensive development, however, has been going on from tunnels driven in the mountain sides as nearly as possible at the cañon level, one on the west side and one upon the east side of the cañon, which traverses the full length of the property.

No development has been done below the level of these tunnels, but above them mining has been planned in such a way as to permit the extraction of ore by the caving system until it is deemed expedient to apply other and more economical means. It has not been found necessary to waste any material from the mine, nearly everything encountered in de-

velopment having been of a profitable working grade. Certain drill-holes have been made at various points in the property to a depth below the tunnel level, which indicate continuous orebodies of a profitable grade at much greater depth, and while the developments have been extensive, they have not disclosed the limits of the orebody.

The average assay of 6000 samples is so nearly identical with that of the average ore sent to the mill during the year that it can be stated that a fair average value of all ore developed is 1.98 per cent. copper, and the average assay value of gold and silver is 0.016 oz. and 0.15 oz. per ton, respectively.

The extraction of ore by stoping did not begin until November, 1904. Prior to that time practically all the tonnage treated by the mill was that extracted in the course of development.

Various and extensive experiments necessarily interfered with the continuous operation of the milling plant and reduced the number of tons treated considerably below the capacity of the plant; but all of these experiments and the resulting expenditures have been made with the idea of directing the construction of the new plant. Judging from results already obtained at the present mill, the first unit of 3000 tons per day of the new mill will produce copper at 8c. per pound.

The company has authorized an issue of $3,000,000 in 6 per cent. convertible bonds, the proceeds of which will be expended in the retirement, on July 1, 1906, of the entire present outstanding bonds, amounting to $750,000 par value, bearing interest at 7 per cent., and in the construction of a mill capable of treating 6000 tons of crude ore per day. The first unit of this mill, having a capacity of approximately 3000 tons per day, will be ready for operation by November, 1906, and the second unit will be installed thereafter as soon as practicable.

The present experimental plant, located in Bingham Cañon, together with the first unit, should result in a production of copper between 30 and 35 million pounds per annum, which will be correspondingly increased upon the completion of the second unit.

Notwithstanding that the mill was conducted as an experimental plant during the year ending June 30, 1905, and large outlays were made for mining developments, the profits in operation amounted to $198,738, from which taxes and interest on bonds outstanding, which sums total $56,250, were provided, leaving a net gain of $142,488 for the year.

The Starless group was purchased by E. A. Wall, former owner of the ground now operated by the Utah Copper Company, and it will form the basis for a similar enterprise, the group being located within the porphyry zone.

The United States Mining Company maintained about the usual output

from its Bingham mines, but it drew heavier than ever before upon its Centennial Eureka mine in Tintic.

The Bingham Consolidated obtained from its Commercial mine in Bingham cañon between 250 and 350 tons per day, from the Dalton & Lark between 150 and 250 tons according to smelter requirements, and an average of 70 tons per day from the Eagle and Blue Bell mines of Tintic. The production of the Bingham smelter in 1905 comprised 14,396,269 ℔. of copper.

The Yampa Smelting Company, owned by the Tintic Mining and Development Company, added considerable equipment, and corrected many of the imperfections in its plant in lower Bingham. This company handles only the ore of the Yampa mine and, with the new equipment now being installed, will have facilities for the treatment of about 1000 tons of ore per 24 hours.

The New England Gold and Copper Company conducted a successful development campaign, and has kept its 50-ton mill in operation steadily.

The Bingham Central Mining Company has opened some good orebodies and is expected to be ready to produce in 1906. A corporation known as the Utah Development Company has been organized to operate the Red Wing properties in the Markham Gulch division of Bingham. Four distinct orebodies were opened and shipments of from 500 to 600 tons monthly maintained.

The Utah Consolidated increased its ore reserves and reduced its cost of production. The output was about 750 tons per day, approximately. The copper production in 1905 amounted to 17,264,474 ℔.

Vermont.—There was no production of copper in this State in 1905, but plans for production were inaugurated and the State should figure as a producer of copper in 1906. Magnetic separation plants are being installed at the Elizabeth mine, at South Strafford, and at the Pike Hill mine, at Corinth. The plan is to enrich the ore by removal of the pyrrhotite. The Westinghouse enterprise at Copperfield was abandoned and the plant dismantled.

Wyoming.—The copper production of this State continued to be derived chiefly from the Encampment district, where the Penn-Wyoming Copper Company is the principal producer. The operations of this company were extended during 1905. The mines at Hecla, near Cheyenne, have not yet proved to be of much consequence.

Copper Mining in Foreign Countries.

Since 1903, Mexico has displaced Spain-Portugal as the most important copper producing country of the world, and in all probability it will continue to gain in position, inasmuch as its copper production is increasing

rapidly, while that of Spain and Portugal is practically at a standstill. In 1905, the United States, Mexico, Spain and Portugal, Australia, Chile, Japan, Germany, Canada, Russia and Peru, which ranked in the order stated, produced upward of 95 per cent. of the total output of copper. Outside of these countries there are no known deposits of copper which promise to attain speedy importance. There are believed to be extensive deposits in Africa, especially in the Congo Free State. The latter are to be examined scientifically by European engineers during the present year. It will be, in any case, many years before these can be brought to the stage of high production. It is worthy to mention, however, that the gradual construction of the Cape to Cairo railway will doubtless develop important mining regions in this continent.

Algeria.—No development of the deposit near Ain Sefra, mentioned in Vol. XII, has been reported.

Australia.—The most important producers of copper in this continent, with which for purposes of review we class Tasmania, are the Mt. Lyell and Wallaroo & Moonta companies, the former in Tasmania, the latter in South Australia. A new producer, which will enter the list in 1906 and promises to be of great importance, is the Mt. Morgan Gold Mining Company, of Queensland. The total production of copper in Australia in 1905, however, showed only a small increase over 1904.

Nevertheless, the copper mining industry is in a flourishing condition, and the outlook is encouraging. The production by the Mount Lyell company was on about the same scale as during 1904, and amounted to some 8300 tons. The profits, however, showed a decided increase, and amounted to $1,605,000. The ore yields under 2.25 per cent. copper, but, including gold and silver contents, is worth about $10 a ton, gross. The bodies of ore opened up at the 600 and 700-ft. levels have added greatly to the life of the mine.

In South Australia, the mines on the Yorke peninsula have been in continuous operation, and the output may be set down, approximately, as 6000 tons.

In New South Wales, the yield from the mines at Cobar, Nymagee, and Burraga was well in excess of that for 1904, and the general prospects are distinctly favorable to the output being still further augmented.

In Queensland, the main supplies were, as usual, drawn from the mines controlled by the Chillagoe company, but several mines of much promise, especially the "O. K.," were opened up during the year. The treatment by the Mount Morgan company of the extensive bodies of gold-copper ore should result in a valuable addition to the copper production of Queensland during 1906.

The discovery in the Mount Morgan mine, previously known only as a gold mine, was made by diamond drilling, which revealed an immense body

of auriferous copper ore, previously unsuspected, in the lower levels of the mine. Plans were immediately undertaken for its development.

In the report of the Mount Morgan company for the half year ending Nov. 30, 1905, it was stated that operations in the mine in connection with the auriferous copper ore were confined to making all the openings necessary for mining it. The workings extended for the purpose of extracting what has been already found proved that a greater quantity exists than was formerly estimated. The copper reduction works will be in operation in February, 1906. It will comprise three blast furnaces, and a converter plant.

The Wallaroo and Moonta mines have been described by L. Hancock[1], general manager of the company. Mining for copper on Yorke Peninsula dates back to just before 1860. The Wallaroo mines are about six miles east, and the Moonta mines about 12 miles south of Port Wallaroo. Both properties are connected by Government railway with the Port, where the company's smelting works are located.

At the Wallaroo mines the lodes traverse a metamorphic schistose rock, with a direction approximately east and west. The various ore-bearing strata at the Moonta mines are composed chiefly of porphyritic rock, and the lodes (speaking generally) bear north and south.

The dressed ore from Wallaroo has, throughout recent times, averaged about 11 per cent., that from Moonta about 20 per cent. of copper, excepting that in later years it has been 2 or 3 per cent. lower. For a long time the vein stuff as raised to surface at both properties has contained on the average from 3 to 4 per cent. copper.

Recently immense accumulations of tailings and slimes from mechanical dressings have been subjected to hydrometallurgical treatment, affording good profits.

For about 30 years the Wallaroo mines and the smelting works were one concern, while the Moonta mines were worked independently, selling their ore to the Wallaroo company. The latter, being a private company, published no records; but from what information is available it would appear that during its separate existence £2,229,096 of copper was extracted, besides nearly £339,000 produced from purchased ores. Apparently about £430,254 was distributed in dividends.

These copper values do not include those from the Moonta company, whose published statements show that £5,396,146 worth of copper was raised, of which £1,168,000 was disbursed among shareholders. The Moonta mines have the distinction of being the first mining company in Australia to pay in dividends a total of £1,000,000, notwithstanding that the rich gold reefs of Victoria had been operated for years before the Moonta was discovered.

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[1]Abstract of a contribution in "Review of Mining Operations in South Australia" in 1905.

Since 1889, when the Wallaroo mines, with their smelting works, and Moonta mines became one concern, copper worth about £4,281,342 has been produced, of which £224,000 has been distributed in dividends. In nearly 45 years these mines have raised and extracted about £12,245,554 worth of copper, and paid £1,822,254 as dividends.

At the smelting works, in addition to the high-class copper of the well-known "Wallaroo" brand, sulphuric acid and bluestone are manufactured.

Early in 1904 a fire in the main shaft at Wallaroo mines completely destroyed the upper portion, and rendered useless the pumping appliances to a depth of 2000 ft. There was grave danger that the deep workings would be lost through the flux of water, but by strenuous exertions, and despite great difficulties, temporary pumping appliances were installed, driven by compressed air. These are now being superseded by an up-to-date electric scheme; also an extensive central steam power and compressed-air plant is nearing completion. A new shaft has been sunk 840 ft., to join the old one where the underlay passed to the vertical.

When the new surface and underground plant is completed Wallaroo mines should be re-established on modern and profitable lines, and as the deepest levels (1800 ft. to 2000 ft.) show no diminution in value as compared to those of less depth, the prospect of continued operations for years ahead is promising.

As further illustrating the operations at the mines, some additional statistics are appended, which, although not absolutely certified, are believed to be practically correct: Total dressed ore produced from the combined mines, 1860 to 1904, inclusive, 1,540,180 tons; average copper contents, 15½ per cent., representing in fine copper 235,630 tons; average yearly production of ore, 34,226 tons; maximum output in one year, 40,222 tons; total expenditure, £10,423,300; average annual expenditure, £231,629; average cost of each ton of dressed ore, £6 15s. 4d.; number of employees, June 1905, 2260; maximum number of employees, 2600; total amount of dividends, £1,822,254.

Brazil.—The only copper mine operated in this country is the Gamaquan, in Rio Grande do Sul. Copper deposits on the Paco Alegre plantation, in the Guraca region, Bahia, were examined during 1905. These deposits are situated in the vicinity of the Central railway of Bahia. There are said to be other deposits in the same district. Copper is also reported to exist in the States of Maranho and Seara.

Canada.—The production of copper in Canada in 1905 was 47,597,502 lb., against 42,970,594 lb. in 1904, these being the statistics (subject to revision) of the Dominion Geological Survey. There are several copper producing districts, the most important being Sudbury, Ont., and the Boundary district, B. C.; the latter is by far the most productive, and

promises to increase largely in output. Copper producing districts of less importance occur in Ontario and on the Pacific coast.

In the Boundary district there are three large producers, these being the Granby Mining Company, British Columbia Copper Company, and Dominion Copper Company, the last being a reorganization of the old Montreal & Boston, which was effected in 1905.

The Granby company has an enormous deposit of sulphide ore, very low in grade, but so situated that mining can be done very cheaply, and of such composition (self-fluxing) that smelting can be done at an extraordinarily low cost. The company is now paying dividends, and is making preparations to increase its output largely. It is adding another converter stand to its plant, which will give it a converter capacity of about 3,000,000 ℔. per month. At present the company is making a little over 2,000,000 ℔. per month. The company is also preparing to remodel one of its smelting furnaces, with a view to increase its capacity from 300 tons per day to 450 tons. The smelting works are now treating about 80,000 tons of ore per month.

The British Columbia Copper Company operated at a small profit in 1905 under the favorable conditions as to price of copper which prevailed.

The Tyee Copper Company, of Vancouver Island, made a diminished output in 1905, owing to the exhaustion of the large orebody which previously had been worked. Toward the end of the year, however, it was reported that ore had been rediscovered. The Britannia Copper Company, of Howe Sound, Vancouver Island, completed a new smelting plant in 1905 and began production toward the end of the year.

The Vancouver smelters received considerable ore from Alaska, from which source the offerings are increasing. Their product is shipped to the Pacific coast smelters of the United States, for refining. The output of the smelters of the Boundary district, which is made in the form of blister copper, is shipped to New York for refining.

The falling off in shipments from the Tyee mine, Vancouver Island, and from Texada Island mines, reduced the copper output of the Coast. Rossland also made a small loss, but the Boundary district more than made up for these losses, the largest part of its increase having come from the Granby mines, at Phœnix. The total production of copper in British Columbia in 1905 was 37,692,251 ℔., against 35,710,128 ℔. in 1904.

Chile.—At Antofogasta, the Guanaco, once a gold mine, is now being worked for copper, which appeared at a depth of 328 ft.; the bottom workings (820 ft.) are still in iron and copper pyrites. Several deep copper mines are being worked in Chile. In the department of Chañaral the Fronton mine has attained a depth of 1837 ft., the bottom workings still being in good ore. One vein of pyrites, about 6½ ft. thick, yields from 7 to 8 per cent. copper; while a second vein, about 2½ ft. thick, yields from 10 to 11

per cent. The Descubridora de Carrizalillo mine, in the same region, has a depth of 2132 ft. In Los Pazos district two large veins of oxidized ores. varying from 26 to 65 ft. in thickness, are being worked in shallow depths, In the department of Copiapo, La Dulcinea, worked by the Copiapo Mining Company, is the deepest mine and the largest producer in the whole of Chile. The vertical depth amounts to nearly 2624 ft., while longitudinally it has been worked for 1640 ft. Copper pyrites was met at a depth of 656 ft. The vein, 5 to 8 ft. thick, occurs in a granite rock, the ore, chiefly copper pyrites, averaging about 18 per cent. of metal. In La Serena department the veins of La Higuera district occur in dioritic rock, the gangue consisting of calcite and asbestos, termed *piedrapalo* by the miners. The Brilladoı and San Antonto mines, to the south of the same department, are about 1800 ft. deep. The famous Tamaya district, in the department of Ovalle, which formerly was the largest copper producer of Chile, is at present worked only on a small scale. The Rosario mine has a depth of about 1900 ft.

Germany.—The copper production of Germany showed a small increase in 1905. Nearly all of the copper output of domestic origin in this country is derived from the Mansfeld mines.

Japan.—There was a small increase in the copper production of Japan in 1905, but owing to the war which was in progress up to the middle of the year, and the general disturbance of industrial conditions, nothing more was to have been expected. It is believed that the exploitation of the mines will now be undertaken on an increasing scale, which will early lead to a larger output.

Mexico.—This country is now the second largest copper producer of the world. Its production in 1905 was 144,350,962 lb., this figure being arrived at by adding the production of the Boleo company to the net imports of copper from Mexico into the United States. The latter amounted to 121,536,582 lb., of which 28,890,239 lb. was copper in ore, and the remainder was blister copper, imported for refining. Some copper is shipped in the form of matte from El Paso, Tex., to Aguascalientes, Mexico, and brought back to the United States as blister copper. Allowance is made for this in computing the net importation.

Of the production in 1905, the Greene Consolidated Copper Company, of Cananea, Sonora, produced 62,189,357 lb., the Moctezuma Copper Company of Nacozari, Sonora, 10,009,781 lb., and the Boleo company of Lower California, 22,800,000 lb. It will be observed, therefore, that the State of Sonora takes the lead in copper production in Mexico. Outside of this State, important producers are the Tezuitlan and Mazapil companies, the former in Puebla, the latter in Zacatecas.

In all probability there will be a further increase in the production of Sonora in 1906, inasmuch as many small companies, which were making preparations in 1905, ought to reach the producing stage in 1906. Among

these are the Transvaal Copper Company, of Cumpas, Sonora; the Ohio-Mexican Copper Company, of Caborca; and the Cieneguita Copper Company, of Sahuaripa. Elsewhere in Mexico there is also activity in copper mining.

The mines of the Greene Consolidated Copper Company have increased their output, and are now mining over 3000 tons of ore daily. This company has accomplished very much during the four years it has been operating. Over 33 miles of underground workings, shafts, winzes and levels have been driven, not including 8000 ft. of diamond drilling. The average extraction from the ore now handled is between 3.5 and 4 per cent. of copper. The production of the Cananea mines, from commencement of operations in 1901 to July, 1905, was 86,375 tons of copper.

THE WORLD'S COPPER PRODUCTION, 1902–1905. (*a*)

Countries.	1902.		1903,		1904,		1905.	
	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.
Argentina	240	244	135	137	155	157	155	157
Australasia	26,640	29,098	29,000	29,464	34,160	34,706	36,560	37,145
Austria-Hungary	1,600	1,626	1,385	1,407	1,450	1,473	1,325	1,346
Bolivia	2,000	2,032	2,000	2,032	2,000	2,032	2,000	2,032
Canada	17,485	17,765	19,055	19,360	19,183	19,490	21,248	21,588
Cape of Good Hope — Cape Co.	2,750	2,794	4,630	4,704	5,475	5,563	5,025	5,105
Cape of Good Hope — Namaqua	1,700	1,727	600	610	2,300	2,337	2,300	2,337
Chile	28,930	29,373	30,930	31,424	30,110	30,592	29,165	29,632
Germany—Total	21,605	21,951	30,714	31,214	29,778	30,262	22,160	22,492
(Mansfeld)	(18,750)	(19,050)	(19,593)	(19,810)	(19,265)	(19,578)	(19,565)	(19,878)
Italy	3,370	3,424	3,100	3,150	3,335	3,388	2,950	2,997
Japan	29,775	30,251	3,360	3,1861	34,850	35,408	35,910	36,485
Mexico—Total	35,785	36,357	45,315	46,040	50,945	51,760	68,907	70,010
(Boleo)	(10,785)	(10,958)	(10,315)	(10,480)	(10,945)	(11,120)	(10,178)	(10,341)
Newfoundland	2,860	2,906	2,710	2,753	2,200	2,235	2,280	2,316
Norway	4,565	4,638	5,915	6,010	5,415	5,502	6,305	6,406
Peru	7,580	7,701	7,800	7,925	6,755	6,863	8,625	8'763
Russia	8,675	8,814	10,320	10,485	10,700	10,871	8,700	8,839
Spain-Portugal—Total.	49,790	50,587	49,740	50,536	47,035	47,788	44,810	45,527
Rio Tinto	34,480	35,032	35,810	36,382	33,480	34,016	32,280	32,796
Tharsis	6,710	6,817	6,320	6,421	5,620	5,710	4,345	4,415
Mason & Barry	3,330	3,383	2,430	2,469	2,950	2,997	2,720	2,764
Sevilla	1,545	1,570	1,105	1,123	1,330	1,351	1,280	1,300
Sweden	455	462	455	462	533	542	550	559
Turkey	1,100	1,118	1,400	1,422	950	965	700	711
United Kingdom	480	488	536	544	493	501	500	508
United States	284,284	288,833	316,239	316,239	365,051	370,892	389,122	395,348
Totals	533,699	542,209	592,339	602,832	652,873	663,327	689,277	700,

(*a*) The figures in this table are taken from the annual metal circular of Henry R. Merton & Co., except where returns have been received by THE MINERAL INDUSTRY direct from official sources.

Peru.—The copper production of this country showed an increase in 1905, and it will surely increase further in 1906, the new smelting works of the Cerro de Pasco company having been completed, and put in operation about the end of January, 1906.

The Cerro de Pasco smelter blew in one furnace on Jan. 7, 1906, then stopped three weeks, since when it has been running continuously. This furnace is 56x180 in., and is rated at 500 tons daily capacity. The plant has three furnaces now built, and another on the ground, all of them being of the above dimensions; also four stands of upright converters.

The ore now goes about 6 per cent. copper, this being the second-grade

ore of the mines. The first-grade goes up to 12 per cent. copper. All the ore now treated comes from the Diamante mine, six miles from the smelter, with which it is connected by standard-gage railway. The smelter is using coke from England and coal from Australia, but a railway will soon be completed to the company's coal mines, about 15 miles from Cerro de Pasco. A coal washery is to be installed there. A coking plant of 72 beehive ovens is now being built at the smelter.

The ores are smelted raw, fluxed with 5 to 10 per cent. limestone, using 12 per cent. coke on the charge. The furnaces are charged mechanically, and so far as possible all work throughout the plant is done by machinery. The native labor is poor and is relied upon as little as can be helped.

RIO TINTO COPPER PRODUCTION.

Year.	Pyrites Extracted.				Pyrites Consumed.		Copper Produced at Mines.
	Shipment.	Treatment.	Contents.	Average Copper Contents.	Tons.	Average Copper Contents.	Tons.
1876	189,962	159,196	349,158	1.5%	158,597	1.5%	946
1877	251,360	520,391	771,751	2.375	211,487	2.	2,495
1878	218,818	652,818	871,107	2.78	211,403	2.18	4,184
1879	243,241	663,359	906,600	2.78	236,849	2.45	7,179
1880	277,590	637,567	915,157	2.865	274,210	2.481	8,559
1881	249,098	743,949	993,047	2.75	256,827	2.347	9,466
1882	259,924	688,307	948 231	2.805	272,826	2.401	9,740
1883	313,291	786,682	1,099,973	2,956	288,104	2.387	12,295
1884	312,028	1,057,890	1,369,918	3.234	314,751	2.241	12,668
1885	406,772	944,694	1,351,466	3.102	354,501	2.270	14,593
1886	336,548	1,041,833	1,378,381	3.046	347,024	2.306	15,863
1887	362,796	819,642	1,182,438	3.047	385,842	2.283	17,813
1888	434,316	969,317	1,403,633	2.949	393,149	2.208	18,522
1889	389,943	824,380	1,214,323	2.854	395,081	2.595	18,708
1890	396,349	865,405	1,261,754	2.883	397,875	2.595	19,183
1891	464,027	972,060	1,436,087	2.649	432,532	2.651 1.309	21,227
1892	406,912	995,151	1,402,063	2.819	435,758	2.569 1.465	20,017
1893	477,656	854,346	1, 332,002	2,996	469,339	2.659 1.544	20,887
1894	498,540	888,555	1,387,095	3.027	485,441	2.594 988	20,606
1895	525,195	847,181	1,372,376	2.821	518,560	2.595 986	20,762
1896	591,752	845,580	1,437,332	2.931	549,585	2.529 1.068	20,817
1897	575,733	812,293	1,388,026	2.810	582,540	2.595 967	20,826
1898	644,518	820,862	1,465,380	2.852	618,110	2.600 1.023	20,426
1899	644,271	1,005,573	1,649,844	2.719	636,323	2.511 1.120	20,230
1900	704,803	1,189,701	1,894,504	2.744	665,967	2.553 1.187	21,120
1901	633,949	1,294,827	1,928,776	2.627	641,935	2.680 1.025	21,100
1902	627,967	1,237,322	1.865,289	2.517	595,092	2.342 1.495	21,659
1903	688,919	1,229,619	1,918,538	2.390	667,748	2.320 1.241	21,565
1904	672,344	1,276,475	1,948,819	2.340	663,744	2.105 978	21,218
1905	627,336	1,202,768	1,830,104	2.363	660,724	2.182 1.124	19,530

Spain and Portugal.—The copper production of these countries showed a considerable falling off in 1905, this being due to diminished outputs on the parts of both of the large companies, the Rio Tinto and the Tharsis. The official report of the Rio Tinto company shows that the total quantity

of pyrites extracted in 1905 was 1,830,104 long tons, of which 1,202,768 tons was for local treatment and the remainder for shipment. The copper produced at the mines amounted to 19,530 tons, the quantity being reduced by the drought which prevailed during the year. The copper content of pyrites was 12,750 tons, giving a total production of 32,280 tons for the company during the year. The production statistics of the company for 30 years are given in the table on p. 146.

THE COPPER MARKETS IN 1905.

New York.—There is no doubt that, especially in the earlier part of 1905, large quantities of copper were used for military purposes, due to the Russo-Japanese war.

In addition to fairly large exports to Europe in 1905, there was a large extra demand from the Orient, and there was seldom a month in which the large quantity of 20,000 tons for export was not reached. There was no doubt also a considerable extra consumption on account of new copper coinage in China. During the first half of the year, home consumers covered their wants from hand to mouth only, but when, during the summer, they found more or less difficulty in providing for their requirements, they changed their policy. Up to the end of July the market ruled quiet, but with a very firm undertone, prices keeping about on a level, with an occasional spurt of activity brought about by an increasing demand from abroad.

The year opened with prices for Lake and electrolytic copper at about 15c. per lb., and 14½c. per lb. for casting. These prices, with occasional fluctuations of ⅛ to ⅜c. in an upward direction, remained stationary up to the end of April. Then the demand for the metal from consumers here, as well as abroad, ceased almost entirely, and the few comparatively small lots of copper which were pressed for sale had the effect of lowering prices to the extent of about ½c. per lb.

This condition lasted until the beginning of July, when buyers became alive to the fact that copper for near-by delivery was becoming exceedingly scarce and that premiums were asked and paid for such deliveries. This fact prompted them to change their "hand-to-mouth" policy, and to provide for their supplies up to the end of the year. Such action naturally raised prices to a considerable extent.

The conclusion of peace between Japan and Russia, with its expected era of renewed industrial activity in both countries, also helped matters along, so that by the middle of September 16¼c. per lb. for Lake, 16c. for electrolytic, and 15¾c. for casting copper was reached.

A slight set-back to the firmness and activity in the market was caused by rumors of intended re-sales by Chinese dealers of large quantities of copper previously contracted for. However, these reports were unfounded, and prices continued upward. Large orders were placed by consumers both here and abroad, during the latter part of November, for delivery

well into 1906, with the result that at the end of November Lake copper was quoted at 17½c. per lb., electrolytic 17¼c., and casting copper 17c.

After the placing of these large orders the market became quieter again, but assumed further strength and activity during the first half of December, with large transactions for both Lake and electrolytic at from 18¼ to 19c. per lb. Near-by copper remained exceedingly scarce, and fancy prices were realized for small lots that were wanted immediately. The latter half of the month saw a further increase in activity, but toward the close business became rather dull.

AVERAGE PRICES OF LAKE COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	16.33	16.08	16.55	16.94	16.55	16.00	16.16	16.58	16.69	16.64	16.80	16.88	16.52
1901	16.77	16.90	16.94	16.94	16.94	16.90	16.51	16.50	16.52	16.60	16.60	14.39	16.55
1902	11.322	12.378	12.188	11.986	12.226	12.360	11.923	11.649	11.760	11.722	11.533	11.599	11.887
1903	12.361	12.901	14.752	14.642	14.618	14.212	13.341	13.159	13.345	12.954	12.813	12.084	13.417
1904	12.533	12.245	12.551	13.120	13.000	12.399	01.505	12.468	12.620	13.118	14.456	14.849	12.990
1905	15.128	15.136	15.250	15.045	14.820	14.813	15.005	15.725	15.978	16.332	16.758	18.398	15.699

AVERAGE PRICES OF ELECTROLYTIC COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	15.58	15.78	16.29	16.76	16.34	15.75	15.97	16.35	16.44	16.37	16.40	16.31	16.19
1901	16.25	16.38	16.42	16.43	16.41	16.38	16.31	16.25	16.25	16.25	16.22	13.82	16.11
1902	11.053	12.173	11.882	11.618	11.851	12.110	11.771	11.404	11.480	11.449	11.288	11.430	11.626
1903	12.159	12.778	14.416	14.454	14.435	13.942	13.094	12.962	13.205	12.801	12.617	11.952	13.235
1904	12.410	12.063	12.299	12.923	12.758	12.269	12.380	12.343	12.495	12.993	14.284	14.661	12.823
1905	15.008	15.008	15.125	14.920	14.627	14.673	14.888	15.664	15.965	16.279	16.599	18.328	15.590

London.—The year opened with a very hopeful outlook for copper. There was an early run upon refined sorts, particularly for Continental consumption. Available supplies being quickly absorbed, operators turned their attention to the standard market, wherein three months' warrants stood at £68 15s. The closing prices for the month were £67 15s. to £67 17s. 6d. for cash, and £67 17s. 6d. to £68 for three months.

In February the standard market was depressed during the early part of the month, although the turnover was large; but an improvement was soon prompted by requirements for railway material at home as well as by a revived inquiry from India for manufactured sorts and from China for large quantities of copper. Accordingly, about the middle of the month, cash standard rose steadily from £67 7s. 6d. to £68 6s. 3d., at which the month closed; three months' being £68 15s.

March opened with an increased trade in all branches of industry, and consumers barely supplied. The shipments to China were steadily increasing. In contrast to this activity in refined sorts, the standard market was depressed by bear sales which brought the price down to £67 17s. 6d. for cash, and £68 5s. for three months. The month was still young, however, when covering of bear sales caused an advance to £68 15s. for cash, and £69 2s. 6d. for three months. Thereafter, speculation inclined again to the fall and the month closed at £67 7s. 6d. and £67 15s. respectively.

In April the downward movement continued, prices falling to £66 17s.

6d. and £67 5s. respectively, but by the middle of the month the market had risen to £67 5s. for cash standard, and £67 12s. 6d. for three months. Then followed a relapse, caused primarily by a heavy decline in Rio Tinto shares; bears became increasingly aggressive, until the depression brought forth reduced quotations for Lake brands. Other refined sorts were well maintained, so that the margin between standard and refined was abnormally wide. The month closed with cash standard at £65 17s. 6d. and three months' at £65 18s. 9d.

May opened with further depression, and prices fell to £64 16s. 6d. for cash and £64 17s. 6d. for three months. This was the starting point for an improvement, based on increased demand from consumers whose requirements would brook no delay. The month closed, rather quietly, at £65 2s. 6d. for cash, and £65 5s. for three months.

June opened with improved prospects. The Japanese victory over the Russian fleet served to dissipate the prolonged suspense which had prevailed, and gave hope of an early restoration of peace. Bears hastened to cover their commitments, and had to pay advancing prices up to £66 5s. for three months' warrants. This was followed by a reaction of about £1 per ton. Further retrogression was arrested by the urgent demand for refined copper and a revival of Indian demand for manufactured sorts. By the middle of the month standard stood at £66 for all positions. Toward the close of the month it became increasingly evident that consumption was quite abreast of production and threatening to outrun it. Standard, however, was comparatively neglected and closed at £65 15s. to £65 17s. 6d. for all positions.

July was comparatively uneventful until about the third week, when the signs of expanding trade were manifest on all sides. Manufacturers bought refined sorts boldly, and advanced their own selling prices. Under these conditions standard could but follow, and prices rose rapidly to £68 7s. 6d. for cash, and £68 12s. 6d. for three months.

August found the improved conditions fully established. Before the month was half over spot copper commanded a premium. Trade continued good, and it required nothing sensational to promote the rapid advance in prices which prevailed almost to the end, when as much as £72 10s. was paid for standard deliverable in September. At this point, however, some uneasiness was caused by persistent reports of large accumulations in speculators' hands. The origin of these reports was known to be untrustworthy; but the apprehension was nevertheless widespread, and resulted in forced realizations. Speculation apart, the business in refined sorts was exceedingly active, with little or no abatement in price. The month closed with standard at £70 12s. 6d. for cash, and £70 7s. 6d. for forward delivery.

September opened under the shadow of the persistent rumors which had depressed the market in the concluding days of August. English consu-

mers withdrew their support from the market. Bears were alert to take advantage of the situation. Holders became alarmed, and in many cases followed the lead of the bears; and prices within a few days fell over £5 per ton. This episode will long be remembered as the fruit of a baseless rumor circulated for ulterior ends. Meanwhile legitimate trade remained healthy; the rest of the month saw an improvement, from £67 15s. on Sept. 7 to £70 7s. 6d. on Sept. 30, cash warrants commanding 15s. more.

October opened with confidence generally restored, further shrinkage in the actual stocks of standard copper, and a rapid advance to £71 15s. for spot warrants, prompted by large bear coverings. The declaration of peace, after the long and costly Russo-Japanese war, was an additional incentive to the already enormous trade. After a reaction to £69 17s 6d. on October 12, prices advanced to £72 15s. for October dates. Toward the end of the month the internal troubles in Russia inspired apprehension in financial circles, speculation dwindled. and the month closed with standard at £71 6s. 3d. for early dates, and £70 10s. for three months.

November showed early signs of underlying strength. By the tenth the premium on spot warrants had mounted to £2 per ton. Meanwhile consumers found themselves obliged to throw off all reserve and to cover requirements as best they could; the result being a rapid rise in the price of three months' standard, and a consequent narrowing of the recent wide backwardation. As the month advanced it became evident that the continued activity of the consuming industries, and the gradual but persistent depletion of stocks—all of which had been evident throughout the year—were now producing something approaching a crisis in which holders would command the situation. Prices advanced quickly, and without putting any apparent check on trade in the manufactured article. The month closed at £77 7s. 6d. with three months' warrants at £77 2s. 6d.

December opened with increasing activity, the dearth of the metal having become more apparent. Famine prices were paid for refined sorts. Standard soon followed, and prices rushed up to £81 for near dates, which price was freely paid on Dec. 14. Thereafter the tension abated, as the most pressing requirements were covered; and standard relapsed nearly £2 in the price for early delivery; but the manifest scarcity of refined sorts kept the standard price from drifting much lower. Toward the close operations were less numerous and prices somewhat irregular, the last quotations being £78 17s. 6d. for cash warrants, and £78 10s. for three months.

AVERAGE PRICES OF STANDARD COPPER (G. M. B.'s) IN LONDON.
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1901	71.78	71.17	69.54	69.61	69.60	68.83	67.60	66.34	65.97	64.11	64.51	52.34	66.79
1902	48.43	55.16	53.39	52.79	54.03	53.93	52.89	51.96	52.68	52.18	51.03	50.95	52.46
1903	53.52	57.34	63.85	61.72	61.73	57.30	56.64	58.44	56.82	55.60	56.30	56.36	57.97
1904	57.500	56.500	57.321	58.247	57.321	56.398	57.256	59.952	57.645	60.012	65.085	66.375	58.884
1905	68.262	67.963	68.174	67.017	64.875	65.881	66.887	69.830	69.667	71.406	74.727	78.993	69.465

Progress in the Metallurgy of Copper in 1905.

By L. S. Austin.

The literature of copper smelting continued to be enriched during 1905 by the series of articles on pyrite smelting which appeared in the *Engineering and Mining Journal.* These having been republished in book-form, under the name of "Pyrite Smelting," which is readily available, it is unnecessary to summarize them in this review. A further important contribution to the literature of this subject was the elaborate historical paper of Robert Sticht, presented as his presidential address before the Australasian Institute of Mining Engineers.

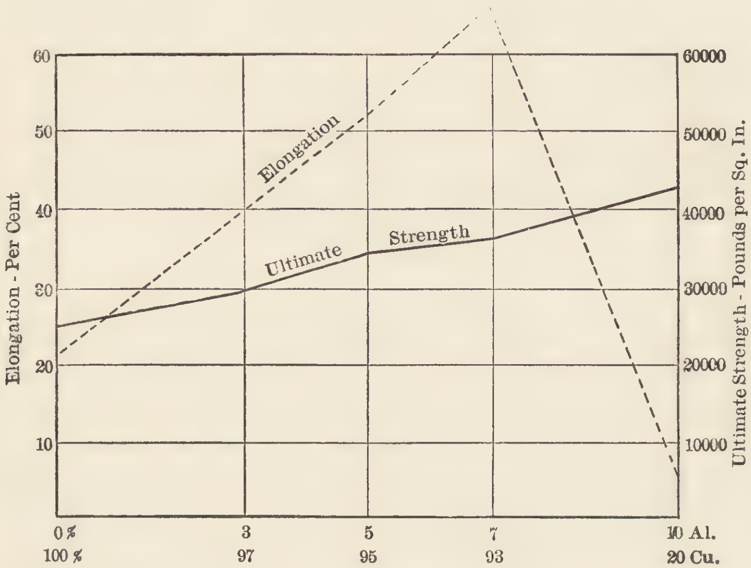


Fig. 1.—Aluminum Bronze, containing up to 10 per cent. Al.

Copper Alloys.

Copper-Aluminum Alloys.[1]—The commercial alloys are of two classes. The first is called aluminum-bronze, and contains not more than 10 per cent. aluminum. The second is called simply aluminum, and carries no more than 3 to 6 per cent. copper.

These bronzes, up to 10 per cent. Al, can be rolled, but do not draw into wire when containing over 7 per cent. The proportions commonly used are, however, 94 to 95 per cent. Cu and 6 to 5 per cent. Al. Another much-used alloy contains 92 to 93 per cent. Cu and 8 to 7 per cent. Al. The aluminum-copper alloys, rich in aluminum, are generally of the type Cu, 2 to 3

[1] *Revue de Metallurgie*, August, 1905, p. 566.

per cent., Al, 98 to 97 per cent. Sometimes, however, we find the type containing Cu, 4 to 5 per cent., Al, 96 to 95 per cent.

Fig. 1 represents graphically the properties of the aluminum-bronzes up to a content of 10 per cent. Al. It indicates that the ultimate

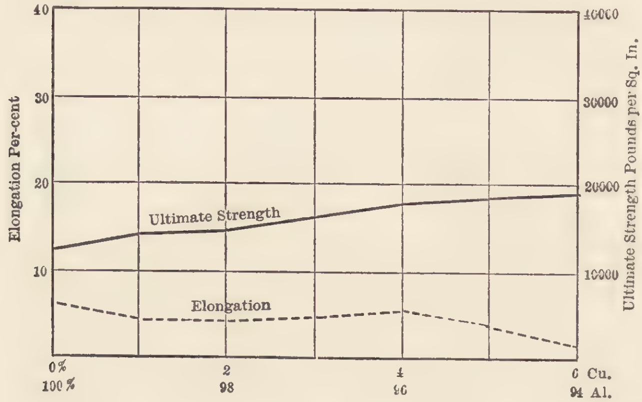


FIG. 2.—ALUMINUM CONTAINING COPPER UP TO 6 PER CENT.

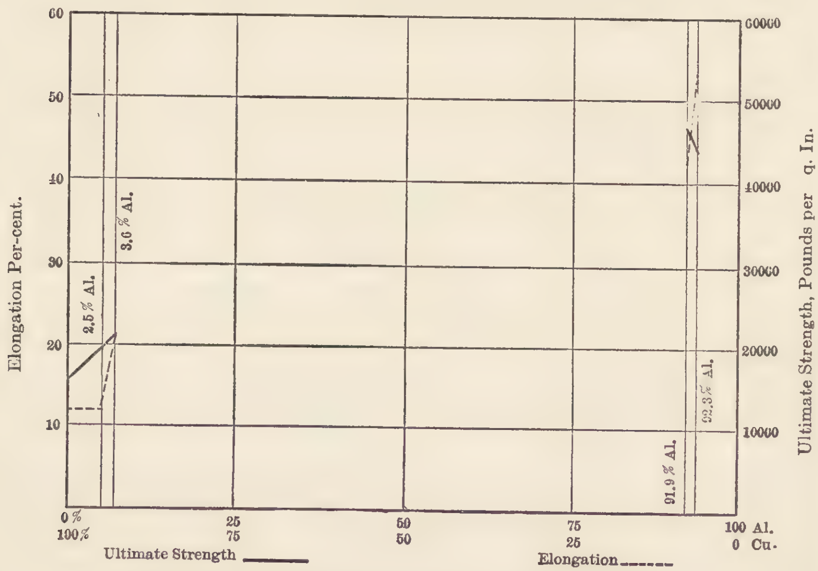


FIG. 3.—COPPER-ALUMINUM ALLOYS, ROLLED AND ANNEALED BARS.

strength and elongation of these alloys steadily increases up to 7 per cent. Al, but, that above this figure, while the ultimate strength increases, the alloy has become hard and brittle.

Fig. 2 gives the corresponding properties of aluminum containing copper

to the extent of 6 per cent. It indicates that up to 4 per cent. the aluminum is improved, but above that the extension at the elastic limit greatly decreases.

Fig. 3 shows the effect of rolling and annealing upon these alloys, and indicates that their strength is nearly doubled after so doing. The elongation is also increased. A series of determinations, made upon annealed bars, shows also that when such bars are heated to 600 deg. C. they have arrived at their maximum of strength and elongation, and have especially improved in the latter. The aluminum-bronzes possess a greenish, golden color, and are used in the manufacture of articles intended to imitate gold alloys. They are also used for springs, and the 10 per cent. alloy for stop-cocks.

Aluminum, containing copper, is used for the covering in of parts of

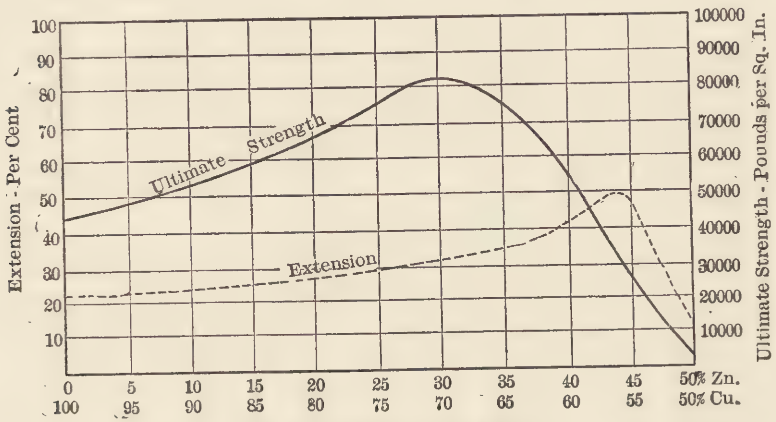


FIG. 4.—ANNEALED DRAWN BRASS.

automobiles, and for places where a light durable metal is needed, as filters, seltzer-water apparatus, vessels to contain milk and wine, cooking utensils, etc. Where it has been used for exposed work, as for cornices, it has not stood well, and, in particular, these alloys do not resist the attacks of the salt-water air.

Complex Aluminum-bronzes.—Attempts have been made to improve the qualities of the aluminum-bronzes by the additions of other elements, such as silicon, boron, copper and cadmium. Experiments on these compounds indicate that silicon and iron play the same role as aluminum in augmenting the ultimate strength, and diminishing the elongation.

Zinc-copper-aluminum Alloy.[2]—This alloy, containing 88 per cent. Zn, 10 per cent. Cu and 2 per cent. Al, stands rough usage better than white metals having a base of lead. The surface of this alloy resembles oxidized silver. The alloy may be cut and polished, and works under the file like

[2] *Engineering and Mining Journal*, LXXX, 638.

gray iron. To prepare it, the copper is melted, and enough zinc added to "fill" it, or prevent its spitting when poured into the main portion of the zinc, which has been melted separately. The mixture is stirred, and when perfectly fluid the aluminum is added and stirred in. It is then cast into plates 1 in. thick. When cool, these plates are broken up and remelted under charcoal, and finally cast into molds.

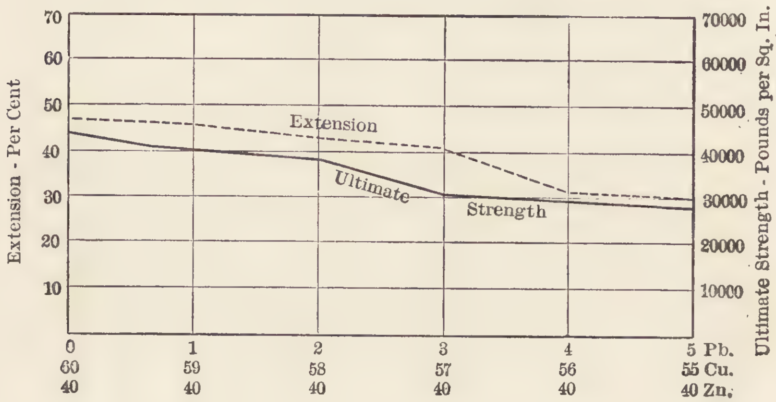


FIG. 5.—LEAD-BEARING BRONZES, ANNEALED DRAWN BARS.

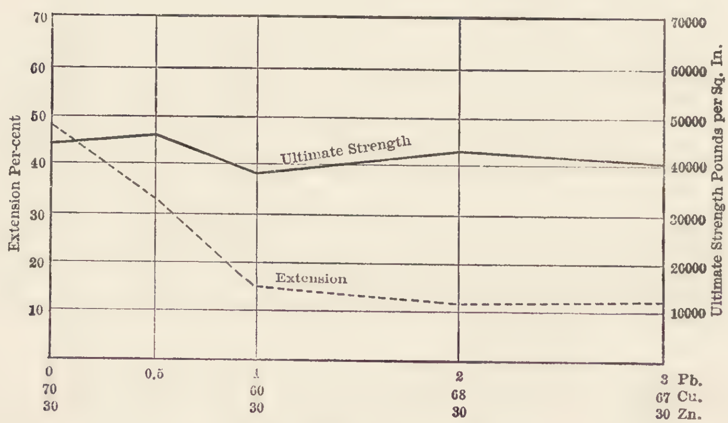


FIG. 6.—LEAD-BEARING BRASS, CAST BARS.

Copper-zinc Alloys.[3]—Fig. 4 gives the ultimate strength in pounds to the square inch, and the percentage elongation of brasses of varying tenor in copper and zinc. The alloys were drawn and annealed before testing.

Figs. 5 and 6 are examples of brasses containing lead. The lead, thus present, facilitates machining, but when over 5 per cent. tends to seg-

[3] M. L. Guillet, *Revue de Metallurgie*, Feb., 1905, p. 97.

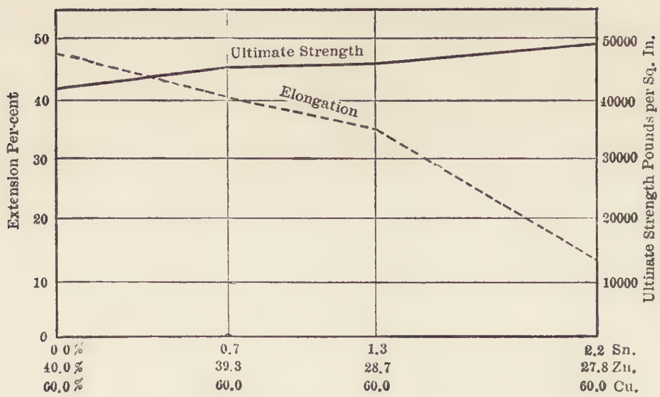


FIG 7.—ANNEALED DRAWN BRASSES CONTAINING TIN.

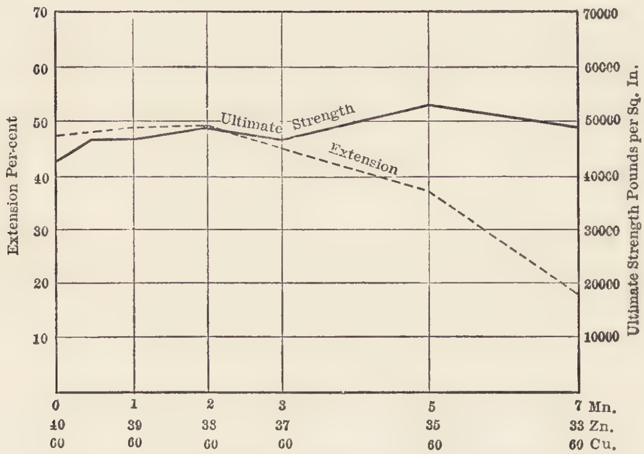


FIG. 8.—MANGANESE BRONZES, ROUGH CAST BARS.

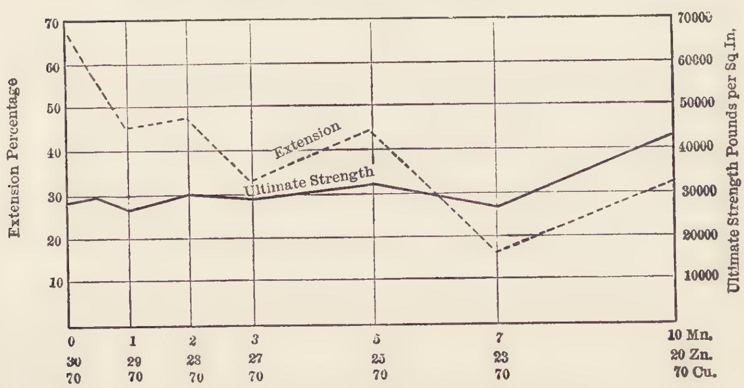


FIG. 9.—MANGANESE BRONZE, ROUGH CAST BARS.

regation, the brass becomes difficult to work, and lead squeezes out. With care, however, it is possible to work brasses of as high as 7 per cent, lead. Above this percentage the lead collects at the bottom of the mold.

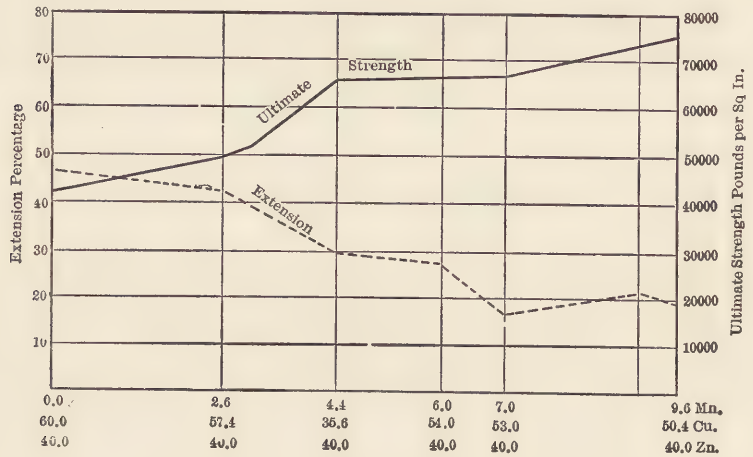


Fig. 10.—Manganese Bronzes, Annealed Drawn Bars.

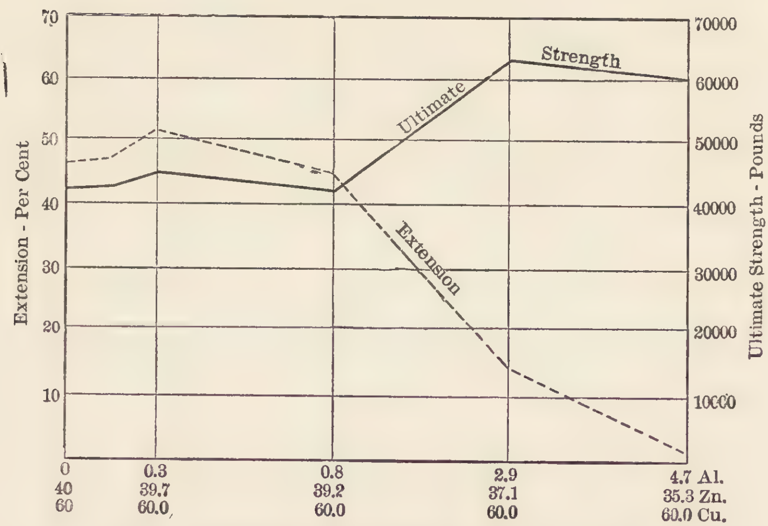


Fig. 11.—Aluminum Bronze, First Type, Rough Cast Bars.

and if the alloy is drawn into wire while hot, the lead spurts out in drops. Thus it can be seen that these alloys do not stand high stresses.

Copper-tin Alloys[4].—The properties of brasses containing tin are shown in Fig. 7. In addition, it has been found that the elastic limit increases

[4] *Loc. cit.*

slightly, and that hardness increases, but the fragility is also greatly augmented. Thus it may be seen that tin can be used only in limited quantities in brasses, and that for cold-worked brass the amount must not

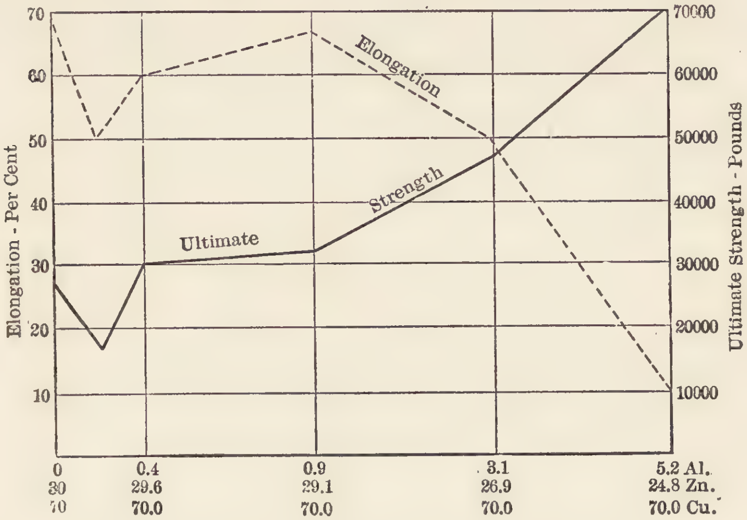


FIG. 12.—ALUMINUM BRONZE, SECOND TYPE, ROUGH CAST BARS.

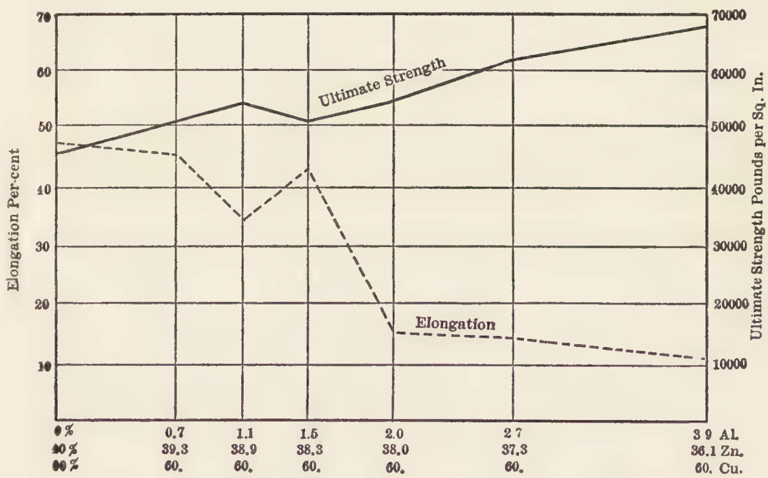
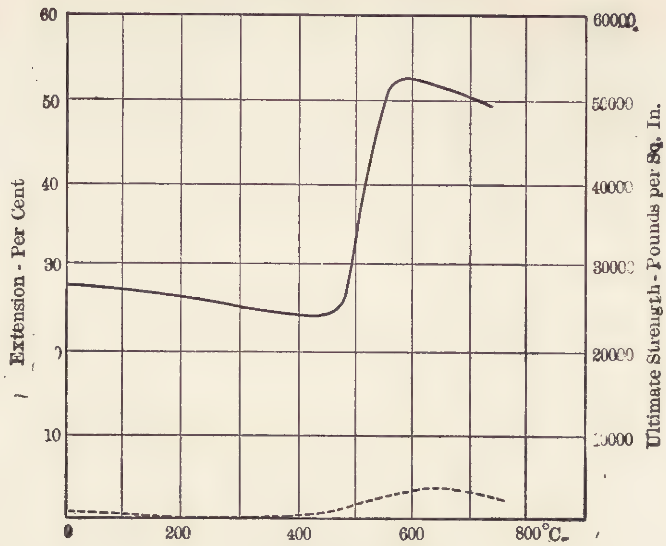
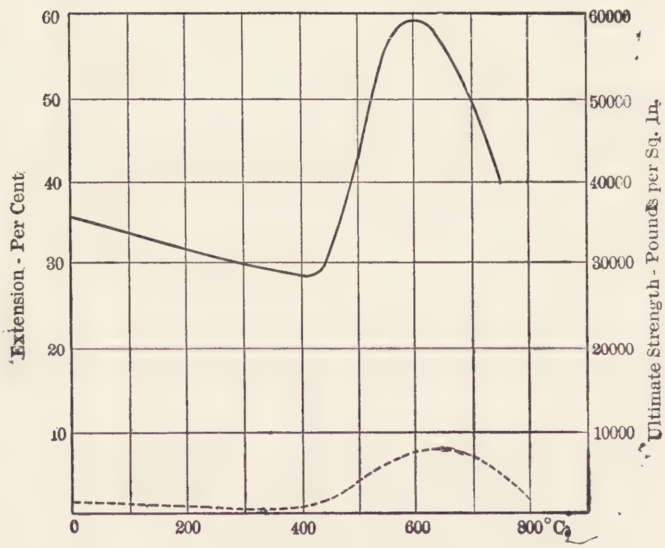


FIG. 13.—ALUMINUM BRONZE, ANNEALED DRAWN BARS.

exceed 2.5 per cent. These alloys have, however, one valuable quality, viz., their resistance to sea-water. Such an alloy contains Cu, 60 to 62 per cent.; Sn, 1 to 1.5 per cent.; and Zn, 39 to 37 per cent. It has been much used in naval construction.



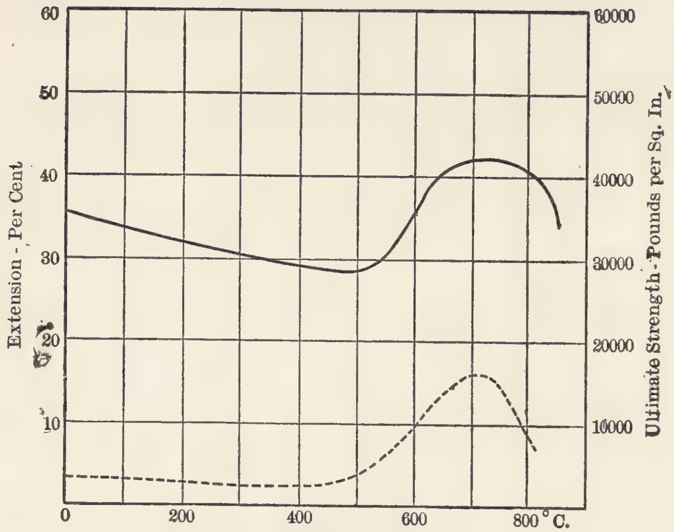
TYPE 1: Cu=79, Sn=21.
FIG. 14.—EFFECT OF TEMPERING BRONZE OF THE TYPE NOTED.



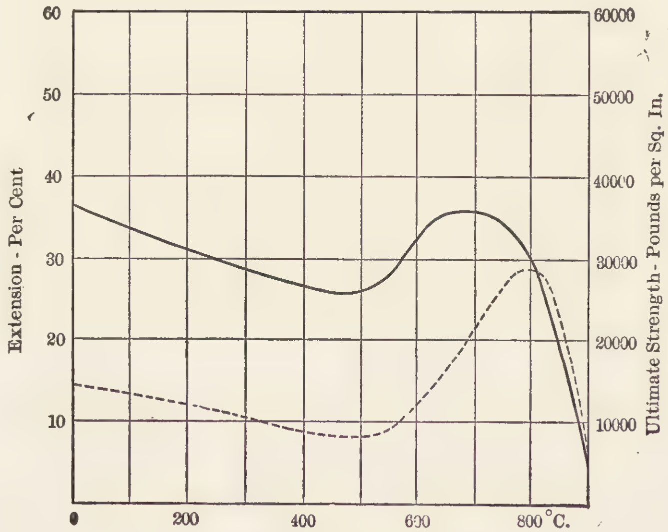
TYPE 2: Cu=84, Sn=16.
FIG. 15.—EFFECT OF TEMPERING BRONZE OF THE TYPE NOTED.

Copper-manganese Alloys[5].—It will be noticed that, in Figs. 8, 9, and 10, manganese has been substituted for zinc, but such substitution makes no change in its micrographic appearance; that is, manganese produces the

[5] *Loc. cit.*



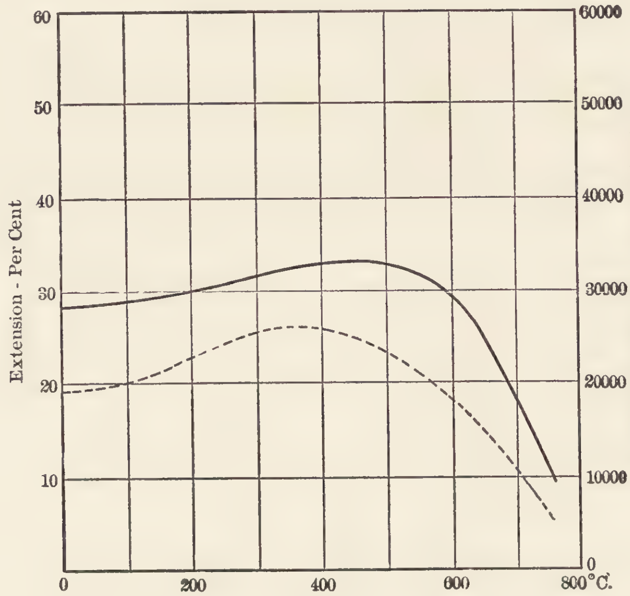
Type 3: Cu=87, Sn=13.
Fig. 16.—Effect of Tempering Bronze of the Type Noted.



Type 4: Cu=91, Sn=9.
Fig. 17.—Effect of Tempering Bronze of the Type Noted.

same effect as zinc in these alloys. Manganese plays a double role. It is a deoxidizer, destroying oxides contained in the melted alloy, and it gives it special mechanical properties. It increases the ultimate strength and elastic limit, but decreases the elongation. Above 4 per cent. Mn the brittleness increases, but the hardness is but little changed.

Aluminum Bronzes.[6]—Aluminum acts as a deoxidizer, as elsewhere in metallurgy. Traces of aluminum gives to a brass of 60 per cent. Cu and 40 per cent. Zn a golden color, which so continues up to 5 per cent. Al. The alloy then becomes rose-colored up to 7 per cent. Al, and viewed under a certain light the color is gray. At 10 per cent. Al the color becomes silver-white. The alloy works well up to 4 per cent. Al, but beyond that it is hard to work; rolling can even then be performed, but not wire-drawing. From 6 per cent. and above, the metal cannot even be rolled. Aluminum, however, acts more strongly than zinc, so that an addition of 1 per cent. Al produces as much effect as 3.5 per cent. Zn. Thus the alloy



TYPE 5: Cu=95, Sn=5.

FIG. 18.—EFFECT OF TEMPERING BRONZE OF THE TYPE NOTED.

Cu 70 per cent., Al 2 per cent. and Zn 28 per cent. has the same properties as a brass of 65 per cent. Cu and $(2\times 3.5)+28=35$ per cent. Zn. It will be noticed from Figs. 11, 12, and 13 that these bronzes are much stronger than the ordinary brasses. Those of 10 per cent. Al, however, are so hard that they cannot be worked.

Tempering of Bronzes.—The copper-tin alloys of Figs. 14, 15, 16, 17 and 18 were heated to the indicated temperatures and suddenly cooled by plunging into water. It will be seen that, for the bronzes containing more than 92 per cent. Cu, the ultimate strength and elongation increases

[6] *Loc. cit.*

somewhat when tempered at from 500 to 600 deg. C. The alloys which contain less than 92 per cent. Cu increase rapidly in tensile strength and in elongation for temperings above 500 deg., and these properties

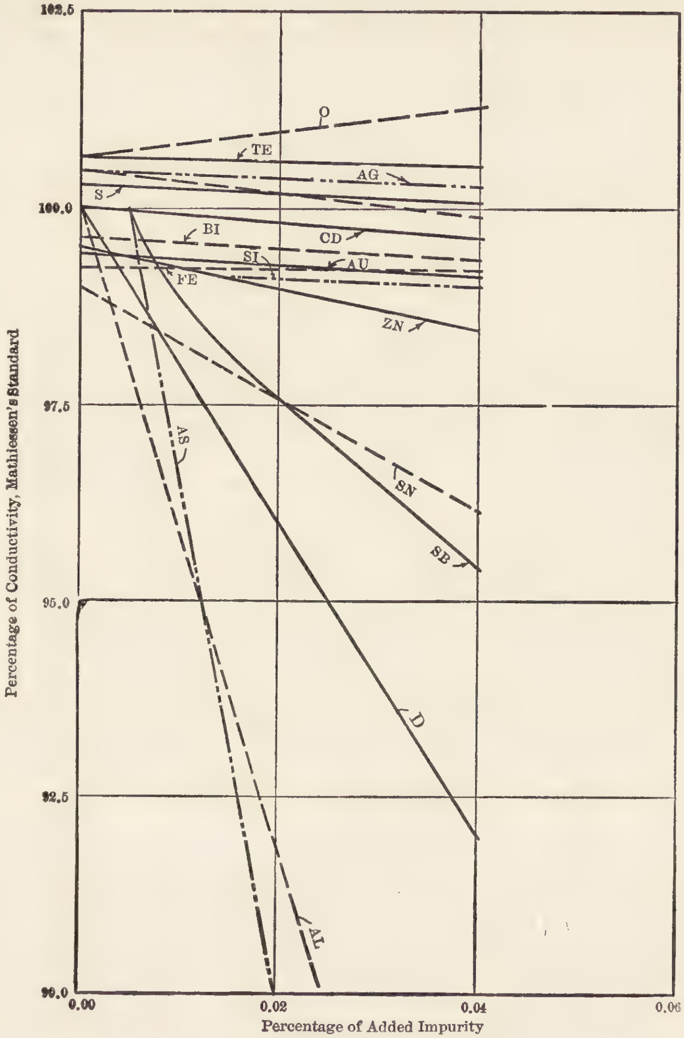


Fig. 19.—Effect of Small Percentages of Impurities on the Conductivity of Copper.

reach their maximum when the alloy has been quenched at 600 deg. C.

Properties of Copper.

Effect of Impurities on the Conductivity of Copper[7].—Small known amounts

[7] *Trans.* American Institute of Mining Engineers, XXXVI, 559.

of the metal or element were added to pure copper, and the resultant decrease in conductivity observed.

Fig. 19, collated from these results, shows the comparative effect up to 0.04 per cent. of the specified impurities upon the conductivity of the copper. (In the original paper the results are carried out much farther.) It will be noted that the elements arrange themselves in two groups, one group having comparatively little effect upon the conductivity when in traces. Indeed, in the case of oxygen, such amounts are rather beneficial, probably as oxidizing other traces also present. The second group, which forms useful alloys with copper, even when added in comparatively small quantity, are from an electrical standpoint the worst ones which can occur. On the other hand the elements of the first group, which tend to make copper brittle, have comparatively little effect upon conductivity.

Electrical Conductivity of Native Copper.—A sample taken from the most compact portion of mass-copper, separating it by means of planing and cutting, then rolling and drawing it into wire of 0.104 in. diameter and annealing, gave a conductivity of 102.5 Mathiesson standard. Cathode copper, carefully deposited with a low current, and prepared in a similar manner, gave just as high a conductivity. Copper, in either of these forms, has a more crystalline structure than even the best Lake copper, whose conductivity approaches 100 Mathiesson standard.

Properties of the Best Grades of Lake Copper.—The Calumet and Hecla mines furnish a copper which, on account of its freedom from natural impurities, affords a very satisfactory commercial copper. Such copper by analysis shows 99.89 per cent. Cu; 0.01 per cent. impurities, as FeS and As; 0.1 per cent. O. Its conductivity, when drawn to 0.104 diameter (about No. 12 Birmingham gage), is 99.5 to 100.0 Mathiesson standard.

Smelting Practice.

Smelting and Refining at the Works of the Michigan Smelting Company[8]. —The works (at Houghton, Mich.) are designed to treat the concentrate or "mineral" produced from the mills of the South Range mines of Lake Superior. The "mineral" is the result both of hand sorting and of the mechanical concentration of the ore or "stamp rock" from the amygdaloid and conglomerate copper-bearing lodes, in which the copper occurs in metallic form. The hand sorting yields two products, viz., "mass copper," varying from the size of a man's head to several tons in weight, and "barrel work," consisting of pieces down to a few pounds in weight, which are packed in oil-barrels for convenient transportation to the smelting works. By mechanical concentration there results headings of almost clean copper (from egg size down to $\frac{3}{8}$ in. diameter) taken from the mortars of the steam

[8] *Engineering and Mining Journal*, LXXXIX, 841

stamps; and several grades of jig and table products varying in size, assaying from 90 down to 30 per cent. copper.

The following table gives a fair idea of the proportions and copper contents of the various grades, being the output of the Copper Range Consolidated mines for four months:

	Percentage.	Dry Weight lb.	Copper %	Copper Contents lb.
Four grades	67.2	11,786,672	68.72	8,101,796
Headings	15.1	2,659,846	90.	2,393,861
Barrel work	6.2	1,089,140	68·	740,615
Mass	11.5	2,016,105	68·	1,370,951
Total	100.0	17,551,763	71.82	12,607,223

The average of the mineral as above given is 72 per cent. metallic copper, the remaining 28 per cent. being the accompanying gangue (amygdaloid rock).

In the operation of the works, as originally planned, the wet mineral as it comes from the mills is received in hopper-bottom cars at the highest point of the works (128 ft. above the main furnace floor), and is weighed and discharged into receiving bins. Thence it is fed by a shaking-shoe mechanism to a 9-A Ruggles-Coles rotary drier of a capacity of 15 tons per hour. The dust is drawn off through an exhaust fan, which also serves to create the draft for the coal fire of the drier. The dust is separated, and caught in conical hoppers (in series) of the Arlington & Curtis type, while the products of combustion escape to the open air. There is said to be no loss of mineral in the operation (?).

The dried mineral is automatically sampled, and discharged into 10-ton cars, which deliver it to the storage bins.[9] There are 10 steel storage bins consisting of cylinders, each 26 ft. 6 in. high by 12 ft. diameter, and having a capacity for 250 tons of mineral. From these bins the mineral is drawn off, as needed, into 10-ton dump cars handled by means of a Jeffrey electric locomotive. It is weighed and dumped into the charging hoppers of the melting furnaces.

In the main furnace building there are two melting and two refining furnaces, with room for a third melting furnace. The melting furnaces have 35x16 ft. hearths, and grates 11x6 ft., or 66 sq. ft. area; and the refining furnaces 23x14 ft. hearths, with fire-boxes having grates 5x6 ft. The present furnaces have a combined capacity of 4500 tons of mineral per month, with a production of 6,300,000 lb. of refined copper. The melting furnaces are charged every 24 hours with 60 to 80 tons of mineral and barrel work, the quantity depending upon the character of the material. As fast as the slag forms in quantities of, say, 5000 lb. it is skimmed into steel molds mounted on a car (five molds to the car), each

[9]At present the drier and sampler are not in use, the mineral being charged to the cars from an upper track for weighing and delivery to the storage bins.

mold holding 1000 lb. This car is handled by an electric locomotive. When the charge has been melted down and skimmed, it is tapped to the adjoining refining furnace (set 7 ft. lower than the melting furnace), along a ganister-lined steel launder. The melting furnace, having been drained, is repaired with sand and ganister as needed, and is then ready for recharging.

Since the refining furnace does not require 24 hours to handle the molten charge of copper, it does its share of melting also, treating the mass, barrel work, scraps and other high-grade material, of which 20,000 to 30,000 lb. is charged each day, after it has been emptied of the preceding charge, and during the interval before the following charge is ready in the melting furnace.

The refining furnace, having been filled, is brought to a good heat and skimmed from slag. The rabbling then begins. For this operation four ¾-in. pipes are introduced at the side doors into the molten copper and supplied with air at 90 lb. pressure to the square inch. This produces an active agitation of the charge, bringing it in contact with the supernatant air, oxidizing the impurities of the copper, and forming copper oxide, which dissolves in the molten copper. This operation lasts about two hours, and brings the metal to the stage of set copper as determined by a sample taken by the refiner, when about 6 per cent. copper oxide remains dissolved in the metal. At this stage there will be the characteristic depression and the single bubble seen at the apex of the depression when the sample is broken. The furnace is again brought to a good heat, and the surface of the bath is skimmed clean. Poling now proceeeds, in which poles of maple or birch are used, 24 ft. long and 6 to 8 in. in diameter at the butt. These are inserted at the working door, and forced into the bath. Charcoal, to the amount of 8 to 10 shovelfuls, is also thrown in. The reducing gases evolved from the wood and the presence of charcoal insures a reducing action upon the cuprous oxide, bringing the copper to tough pitch as determined by the refiner, who takes samples to prove when this point has been reached. At this stage the surface of the sample is flat, the bubble has disappeared, and upon fracture the copper shows a rose color and a silky texture. For the large charges treated, 8 to 12 sticks or poles are needed, the operation taking 1.5 to 2 hours.

Charges of 295,000 lb. have been handled in this furnace; in order to do this, however, the charges from two melting furnaces will be tapped into one refining furnace. The ordinary charge is 220,000 to 230,000 lb.

The copper is cast into ingots, plates and wire bars by a Walker casting-machine, using copper molds. They are warmed with slag the first time, and are sprayed, or painted, with a mixture of lamp-black and benzine at each turn. Charges are poured at the rate of 30,000 to 45,000 lb. per hour, depending upon whether ingots or large cakes are made. The refined cop-

per is handled from the casting-machine direct to industrial flat cars, of 15 tons capacity. These cars are drawn by an electric locomotive to command the standard railroad cars, which take the product to market.

The waste heat from the reverberatory furnace gases is utilized for making steam by being led through Stirling boilers and thence, by underground flues, to a brick-lined steel stack, 15 ft. diameter and 122 ft. high. The base of the stacks being 128 ft. above the furnace floor, a working hight of nearly 250 ft. is secured. The draft at the outlet flue of the melting furnace equals 1.4 in. water; at the working door it is 0.34 in.

The reverberatory slags amount to about 35 per cent. of the copper material received. A representative melting furnace slag is as follows: 40 per cent. SiO_2; 16 per cent. FeO; 11 per cent. CaO; 13 per cent. Al_2O_3; 5 per cent. MnO; 12 to 15 per cent. Cu. The refining furnace slag carries 30 to 35 per cent. Cu. This slag, drawn into molds, as has already been mentioned, is transferred by means of the electric locomotive to the slag crusher. Here a pneumatic hoist is used to seize, lift and dump the slag from the molds into a 15x30 in. rock-breaker, where it is coarsely crushed. The broken slag is raised and transferred to the slag-storage bins by a bucket elevator and an endless bucket conveyor.[10]

The blast furnace is of the standard 38x130-in. copper-matting type, with water-jackets 7 ft. from tuyeres to charge-door. It is placed in a brick building 32x32 ft. From the storage bins the necessary slag, lime rock, iron ore and fuel can be conveniently drawn into the charge buggies. A charge is made up to produce a slag of 40 per cent. SiO_2; 20 per cent. FeO; 16 per cent. CaO; 14 per cent Al_2O_3 and 0.75 per cent. Cu. It is run with 15 per cent. fuel consisting of one-fifth coke and four-fifths hard coal. The brick-lined forehearth is 4 ft. 10 in. by 10 ft. 2 in. The overflowing slag from the forehearth is granulated, as it falls to the waste launder, by a horizontal water-jet. Here it is carried away to the dump by an additional stream of water. The black copper from the furnace is tapped periodically into molds which hold 400 lb. These "cupola blocks" are accumulated, and finally sent to the refining furnace.

The bituminous coal is received in railroad cars upon an overhead track, which are discharged into the main storage bin (of 12,000 tons capacity). This large supply is accumulated to take advantage of the cheap water-freight rates which prevail during the open season of navigation in this region. The coal is drawn from the pile by means of underground drifts, or passage-ways.

Power for the plant is furnished by means of one 12x23x36-in. Nordberg horizontal Corliss engine, belted to a 200 k.w. General Electric a. c. generator. For the blast furnace there is provided a No. 7½ Root blower.

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[10]Both elevator and conveyor have been thrown out, the slag being now raised to the higher level above the slag-bins by means of a skip, whence it is trammed to the desired bin.

One 12x16 and 10x16-in. Sullivan air-compressor supplies the air needed for rabbling the copper and driving pneumatic tools. Electricity, taken from the alternating current generator, is used to drive a motor-generating set of 100 k.w. capacity for supplying the electric cranes and locomotives. The entire plant is served by an industrial railroad having a 36-in. gage track laid with 56-lb. rails. Three Jeffrey electric locomotives are used to handle the cars.

The portion of the main furnace building containing the refining furnaces and casting machines is constructed of steel and brick, 205x160 ft., and is commanded by two 10-ton Shaw electric traveling-cranes for the convenient charging of mass copper and for handling any heavy materials about the furnaces and at the loading-floor.

The melting furnaces at this plant had originally hearth dimensions of 50x18 ft., with a fire-box of 12x6 ft. Much difficulty was experienced in these furnaces, both with the fire-box roof and the sand bottom. The coal used, while of high calorific value, did not have a long flame. It produced a high heat for a distance of 35 ft. through the furnace without undue forcing, but in undertaking to maintain a smelting temperature at the flue end, the roof of the fire-box would last only ten days to two weeks. Several of the sand bottoms, put in in the usual way, also failed and came up, on account of the intense heat at the fire-box end, from the expansion and subsequent contraction in cooling, and from the width of the furnace. While these large furnaces have been successfully used in matte smelting, they have not here been so, the action of molten copper on the sand bottoms being much severer than matte. The furnaces were consequently rebuilt, being reduced in size to 35 ft. x 16 ft. hearth dimensions, with an improved ventilation of the furnace bottom, and, as thus altered, have proved eminently successful, there being no troubles or delays in their operations. They have a capacity in excess of 70 tons mineral in 24 hours.

To avoid the trouble arising from the penetrating nature of the molten copper, which not only went through the two feet of sand bottom, but also into the foundations of the larger furnace, the ventilation of the new furnace was thus provided for. The improved furnace is built upon brick piers covered with iron plates. Upon the plates is laid the bottom, consisting of two layers of silica brick, each 12 in. deep and having 12 in. rise. A brick bottom (rather than a sand bottom) was decided upon, both because of its greater density, and because of its less absorptive power. The ventilation of the hearth, due to the circulation of the air below it, also adds greatly to its durability.

Matte Smelting at Rapid City, S. D.[11]—The plant of the National Smelting Company smelts dry silicious ores of the northern Black Hills,

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[11]*Trans.* American Institute of Mining Engineers, Vol. XXXV, p. 326.

concentrating the gold and silver in a matte of low copper percentage, which is shipped for farther treatment to Omaha, Neb., and to Denver, Colo. There are two furnaces, respectively 144x48 in. and 120x36 in., the first having a capacity of 130, and the second of 105 tons of charge daily. The hight of the furnace, from hearth to feed door, is 15 ft., but the effective hight from tuyeres to bottom of the downtake (smelting column) is 9.5 ft. The furnaces are fed by a specially designed bottom-dump charge-car, which runs directly over the throat of the furnace. The larger furnace receives its blast from a No. 8, the smaller one from a No. 7 Green blower. Both blowers are direct-connected, running 130 to 150 r. p. m. at a pressure of 14 to 18 oz. per sq. in.

The typical slag made at the plant is as follows: SiO_2, 50.2 per cent.; FeO, 16.4 per cent.; CaO, No. 28.3 per cent.; Al_2O_3, 4.2 per cent.; Au, 0.01 oz.; and Ag, 0.2 oz. per ton. The slag flows well. It is granulated and discharged to flat cars for railroad ballast. The slag from the settling-pot falls 4 ft., and strikes the water, flowing in a heavy cast-iron gutter of semi-ellipsoidal section, 8 in. wide by 6 in. deep, inclined 3 in. to the foot for the first 10 ft. The gutter then becomes of somewhat larger cross-section and is inclined 1 in. to the foot.

A hot-blast stove, designed to be heated by the waste gases from the furnaces, increased the temperature of the blast barely 66 deg. F. At one time, and with the help of the waste heat from some cupelling furnaces, as high a temperature as 320 deg. F. was attained.

The charge consists of pyritous ore, silicious dry ore, some Montana copper ore and limestone. This charge requires 14 to 18 per cent. of coke, of which two-thirds is an inferior coke from Cambria, Wyo., and one-third is Eastern or Colorado coke.

The composition of Cambria coke is as follows: Fixed carbon, 65.2 per cent.; volatile combustible, 3.9 per cent; ash, 29.9 per cent.; and moisture 0.9 per cent.

The first matte is low in copper (10 to 14 per cent. Cu), and carries 4 to 5 oz. of gold and 6 to 7 oz. silver to the ton. The matte-fall is 4 to 5 per cent. It has been found that it is absolutely essential to have some copper present in the charge in order that the slags shall be clean, a minimum amount being 0.5 per cent. to each ounce of gold per ton present in the charge.

Such a matte-fall, on the charge used, indicates a high desulphurizing action of the furnace, and this amounts ordinarily to 70 to 79 per cent.

The amount of oxidation being difficult to control, the matte-fall varies also. There are occasions when practically no matte is being made, and yet the slag remains clean. This indicates that a metallic iron sow is being produced. When such a condition of affairs comes about, it is usually remedied by charging extra quantities of pyrite, to furnish more

sulphur for the formation of matte. The first matte is generally resmelted twice, the third matte being shipped away. The subjoined table shows the composition of these mattes.

Matte	Copper	Gold	Silver
First	13.5%	4.04 oz.	6.03 oz.
Second	21.5	10.05	15.6
Third	22.6	17.11	21.4

Sows, or hearth accretions, are one of the accompaniments of these smelting operations. The reactions, producing the metallic iron for them, are thus explained: The first equivalent of sulphur is driven off from the pyrite at a comparatively low temperature. Part of the FeS thus formed is then burned to FeO, and the two react on each other, according to the equation:

$$FeS + 2\ FeO = 3\ Fe + SO_2.$$

The sows consist mainly of metallic iron, with intermixed matte and a little slag. An assay of an average sow showed 20.3 oz. Au and 2.4 oz. Ag per ton. They are broken up and fed back to the blast furnace, a little at a time, with pyrite ores. When operating at a high concentration and with lead or zinc in the charge, there are considerable losses in smoke and fume. Especially are these losses greater when the furnace is run low and with a hot top.

Large Blast Furnaces.—The first of the large furnaces at the Washoe works, Anaconda, Mont., was put in operation March 17, 1905. It has hearth dimensions of 612x56 in., or 247 sq. ft. area, and an effective smelting hight of 10 ft. The furnace has 88 tuyeres. An idea of its construction can be obtained by understanding that two furnaces with their length on the same center line had their adjacent end jackets removed, and the space of 21 ft. between them continued to form a single furnace, giving a hearth of 51 ft. long by nearly 6 ft. wide. The forehearths were left, as before, at the middle of the long sides of the original 15 ft. furnaces. These furnaces had 14 tuyeres to the side, to each of which should be added 16 tuyeres occupying the bridged over portions, thus making 44 tuyeres per side, or 88 tuyeres in all. There are 14 doors to the furnace, each operated by a compressed-air cylinder. The charge-cars are run in on tracks parallel to the length of the furnace. The body is tipped by means of an air-lift, and the charge shot into the furnace. The furnace has two open-breast continuous-discharge spouts, two large forehearths and suitable water-jets and waste launders for granulating the slag. The matte is drawn periodically from the forehearth into matte ladles mounted on cars, whence it is handled to the converters. The furnace works with considerable over-fire.

Cleaning Reverberatory Furnace Slags[12].—At the plant of the Boston

[12] *Trans.* American Institute of Mining Engineers, XXXV, 597

& Colorado Smelting Company, Argo, Colo., the slag, produced by the large furnaces (42x19 ft. hearth dimensions), was formerly skimmed, run out into a sand-bed, broken up and sorted into clean and foul slag. Before loading into railroad cars, both classes were roughly broken and sorted, any pieces containing prills of matte being returned to the subsequent charge of the furnace for resmelting, while the rest, called "clean," was loaded and shipped away. Thus, while the metals in the foul slag were

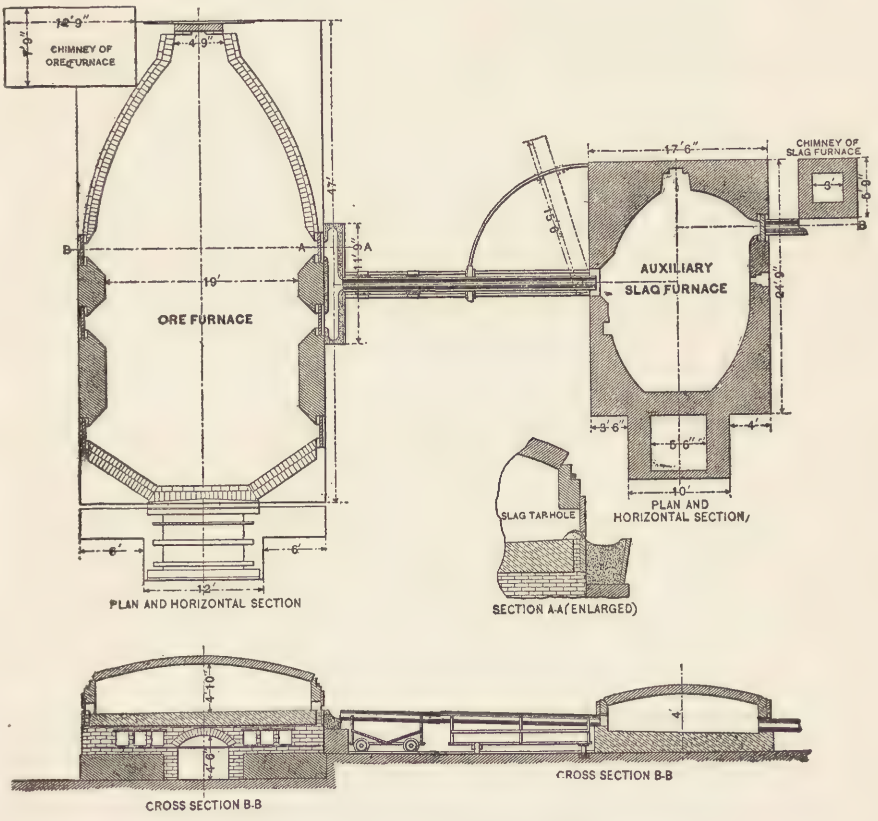

Fig. 20.—Ore and Slag Furnaces.

recovered, it was necessary to go to the expense of rehandling and resmelting in order to do so. Since the slag sorting was done in the open air and under varying weather conditions, the so-called clean slag also contained some which was foul.

The object of the improved method is to treat the entire escaping slag so as to insure that it shall all be cleaned before shipment. The principle is to receive the slag in a smaller reverberatory furnace (20x14 ft.),

where it is treated by the addition of a certain quantity of clean pyrite concentrate, whereby some low-grade matte is made, and precipitated to the hearth of the furnace. The clean supernatant slag is then tapped off, while the matte is removed as it accumulates.

Fig. 20 represents the ore and slag furnaces, both in plan and elevation, as well as the connecting slag-launder by which the slag, as it is skimmed or tapped from the ore-furnace, passes to the slag-furnace. It will be noticed that both the slag and matte-taps are shown in the plan, the hearth sloping so that the matter can be tapped from its lowest point.

The time to be taken in cleaning a charge of slag in the slag furnace is the same as that used in smelting a charge, so that the ore furnace shall never be kept waiting. About 3.5 hours are generally needed for smelting 12 tons of charge, making 84 tons daily. The methods of tapping and cleaning are as follows: The ore furnace being ready for skimming, this is proceeded with at two doors, the skimmers relieving one another at the work. When completed, the chilled launder slag is added to the slag furnace, and the whole fired on for an hour in order to melt it. The side door opposite is then raised, and 1000 ℔. of clean sulphides sprinkled evenly over the slag surface. The door is closed, and the furnace again fired upon for about two hours, when matte, formed from the sulphides, precipitates to the bottom of the furnace. A "bay" or dam, consisting of clay and sand, is formed, as shown in Fig. 20, in the slag tap. When ready to tap, the furnaceman tears the dam down gradually, by means of an iron hook, nearly to the level of the underlying matte, an action which permits of the flow of the slag to the sand molds of the cast-floor. The tap-hole is then built up again ready for the next skimming from the ore furnace. After treating about 20 charges of slag in this manner the slag furnace contains four to five tons of low-grade matte, which is then tapped off by the matte-tap in the ordinary manner, first, however, skimming off the slag lying upon it.

It is to be noticed that in this method the slag is tapped (and not skimmed) from the furnace by lowering the dam. In consequence of the success and ease of operation in this method of removing the slag, it has been adapted to the ore furnace, resulting in a saving of labor, and making the operation an easy and simple one, and doing away with the hot and laborious work. Thus, it is possible to tap several ore furnaces simultaneously, where heretofore it would have been well nigh impossible because of the heat. Also, because of tapping instead of skimming, thousands of dollars are saved in rabble iron annually.

There has resulted from the ore furnaces a cleaner slag than under the old method of skimming, shown by the decreased tenor in the matte produced at the slag furnaces.

Smelting at the Works of the Tennessee Copper Company.—At these

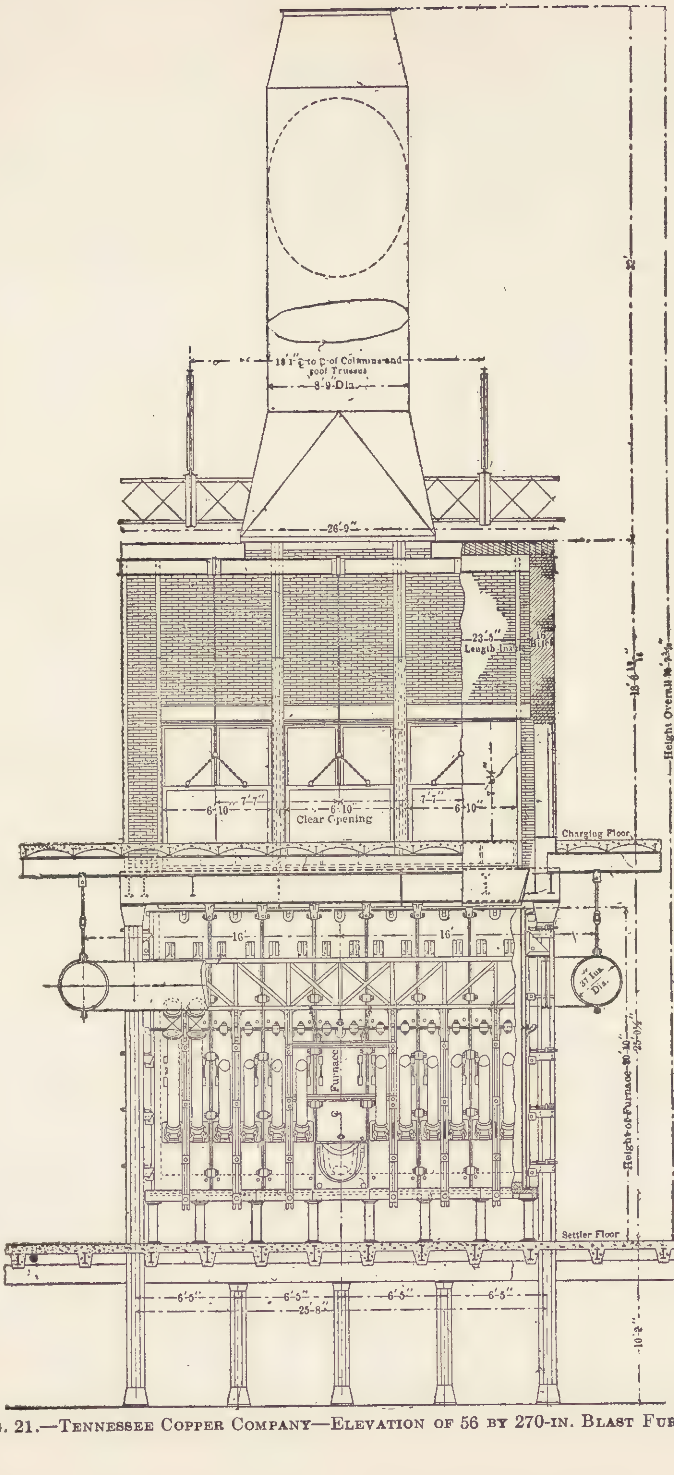

FIG. 21.—TENNESSEE COPPER COMPANY—ELEVATION OF 56 BY 270-IN. BLAST FURNACE

works (Ducktown, Tenn.) there are three furnaces, each 58x180 in., under normal conditions putting through 375 tons of sulphide ore daily, for 27 days in the month, with a consumption of 5 per cent. coke to the charge. They have side boshes of 8 in. and a hight from tuyeres to feed-door of 18 ft., of which 14 ft. is available for smelting column, the top 4 ft. being used for the distribution of the charge. Each furnace has 26 tuyeres, 4 in. diameter of nozzle. The crucible is water-jacketed and has a cast-iron sole-plate. The capacity of a furnace is 375 tons of sulphide ore per day. The charge, consisting of pyrite ore and quartz, is loaded to charge-cars, the quartz being placed on top. The charge-cars dump into the furnace, two on each side. The coke is dumped on the charge-plate and shoveled into the furnace. The blast is furnished by blowing-engines making 68 to 70 revolutions, and furnishing 17,000 actual cubic feet of air per minute. Two of the furnaces are used for treating the raw ore from the mines, and the third for concentrating the first matte from the two furnaces. The ore is a massive sulphide consisting of equal parts pyrite and pyrrhotite, with some chalcopyrite and silica, as in the analysis herewith.

Burra-Burra ore, 2.2 per cent. Cu; 30.0 per cent. S; 3.75 per cent. Fe; 2.0 per cent. Zn; 6.2 per cent. CaO; 1.9 per cent. MgO; 3.9 per cent. Al_2O_3; 10.3 per cent. SiO_2.

London ore, 3.0 per cent. Cu; 21.0 per cent. S; 31.0 per cent. Fe; 0.8 per cent. Zn; 6.1 per cent. CaO; 2.5 per cent. MgO; 4.4 per cent. Al_2O_3; 26.3 per cent. SiO_2.

The ore, as it comes from the mine, is coarsely crushed, not to exceed 6 in. in size. It is stored in the smelter bins and drawn thence, as needed, into charge-cars. An ordinary charge for the furnace consists of 4000 lb. Burra-burra ore, 1000 lb. quartz and 120 lb. (2.5 per cent.) of coke. An other charge would be 5000 lb. London ore, 400 lb. quartz, and 150 lb. (2.8 per cent.) coke. In loading the charge-cars, the sulphide ore is placed beneath the quartz. The coke is fed to the sides and corners of the furnace, while the quartz, being on top of the pyrite ore, tends to go to the center. The coke is regarded rather as needed to prevent the formation of crusts at the sides of the furnace than for a fuel, and, indeed, it is only by careful manipulation that this can be avoided. The furnace must be carefully watched, and at the first sign of crusting must be cleaned off with long bars, or else large pieces of sulphide ore are fed against the crust to prevent it from increasing in size.

Formerly, when these furnaces were operated upon roasted ore, the smelting column was kept at 8 to 9 ft. At present, and for pyritic smelting, the depth of charge will vary from 12 to 14 ft., and, under these conditions, the furnace operates better, and achieves a better elimination of sulphur than with a less depth of charge.

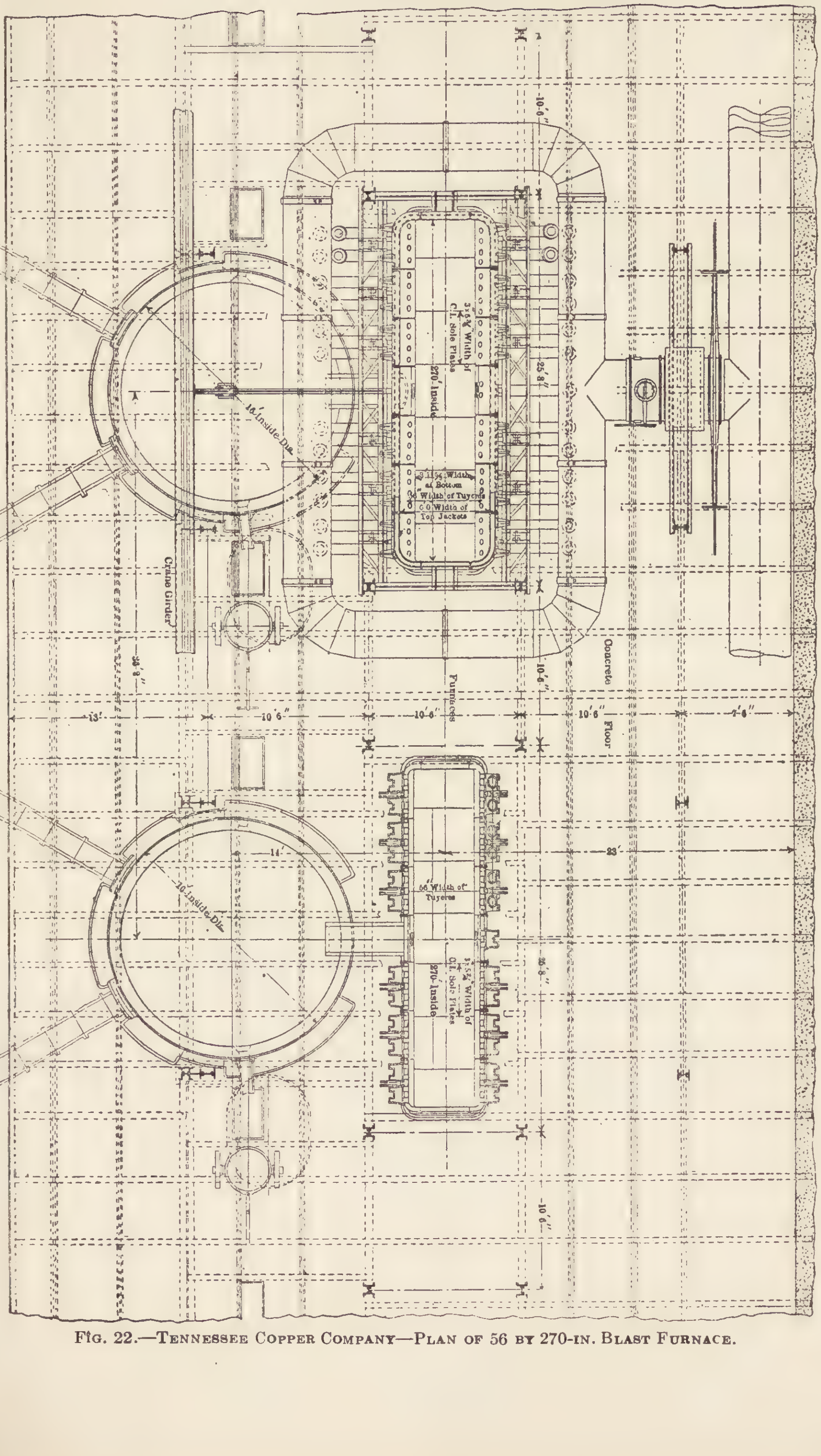

FIG. 22.—TENNESSEE COPPER COMPANY—PLAN OF 56 BY 270-IN. BLAST FURNACE.

The furnace is run with an open breast overflowing into a 16-ft. circular forehearth lined with chrome brick. The slag is removed from the forehearth in slag-cars to the dump. The first matte is also tapped, from time to time, from a spout set near the bottom of the forehearth. The slag-cars run in at a lower level, and take this matte away to a slightly inclined yard, where it is poured into flue-dust beds. After it has solidified and cooled, it is broken up into pieces of about 6 in. size, loaded to railroad cars and brought up to the smelter bins.

The operation in the third, or matte-concentrating furnace, consists in smelting the first matte, together with quartz, converter slag and barrings or cleanings from the blast furnace, so as to produce a suitable grade of converter matte. This furnace is charged occasionally, as it has been found that a matte charge will clean out a crusted-up furnace, and, conversely, that an ore charge will clean out a crusted matte furnace. The matte furnace runs more slowly than the ore furnace, having a capacity of 120 to 180 tons of matte per day. In running one of these furnaces on matte, it is necessary to use a jacketed crucible, otherwise it is not possible to prevent the matte from breaking out.

The first matte contains 11.9 per cent. Cu; 2.56 per cent. S; 54 per cent. Fe, and 1.7 per cent. Zn; and the second or concentrated matte 42.8 per cent. Cu; 24.6 per cent. S; 29.2 per cent. Fe, and 1.0 per cent. Zn.

It is a remarkable fact that, with a given ore and given furnace, the combination tends to make a certain type of slag, even when the proportion of quartz is varied from that above given. At these works the slag tends to come out 40 per cent. SiO_2, and the only change is in the increase or decrease of the grade of the matte. In fact, the way of running the furnace is to vary the quartz charge so as to keep a constant grade of matte. The silica appears to make the first demand on the iron in the charge, the residue being then available for the matte; so that, as the silica increases, so does the grade of the matte. This grade in practice varies from 9 to 15 per cent, but it is aimed to get a matte of 12 per cent.

Herewith are the analyses of the slags of both the first and of the second operations, viz.:

Ore furnace slag: 0.2 per cent. Cu; 0.7 per cent. S; 42.5 per cent. FeO; 1.5 per cent. ZnO; 7.5 per cent. CaO; 2.0 per cent. MgO; 5.5 per cent. Al_2O_3; 40.0 per cent. SiO_2.

Matte furnace slag: 0.6 per cent. Cu; 1.3 per cent. S; 52.6 per cent. FeO; 1.1 per cent. ZnO; 2.6 per cent. CaO; 0.7 per cent. MgO; 4.2 per cent. Al_2O_3; 36.5 per cent. SiO_2.

On page 176 is the result of three runs for the concentration of first matte. First matte of 10 to 12 per cent. was thus raised to 44 to 52 per cent. Cu. As will be noticed, increasing the quantity of quartz increases the grade of the matte.

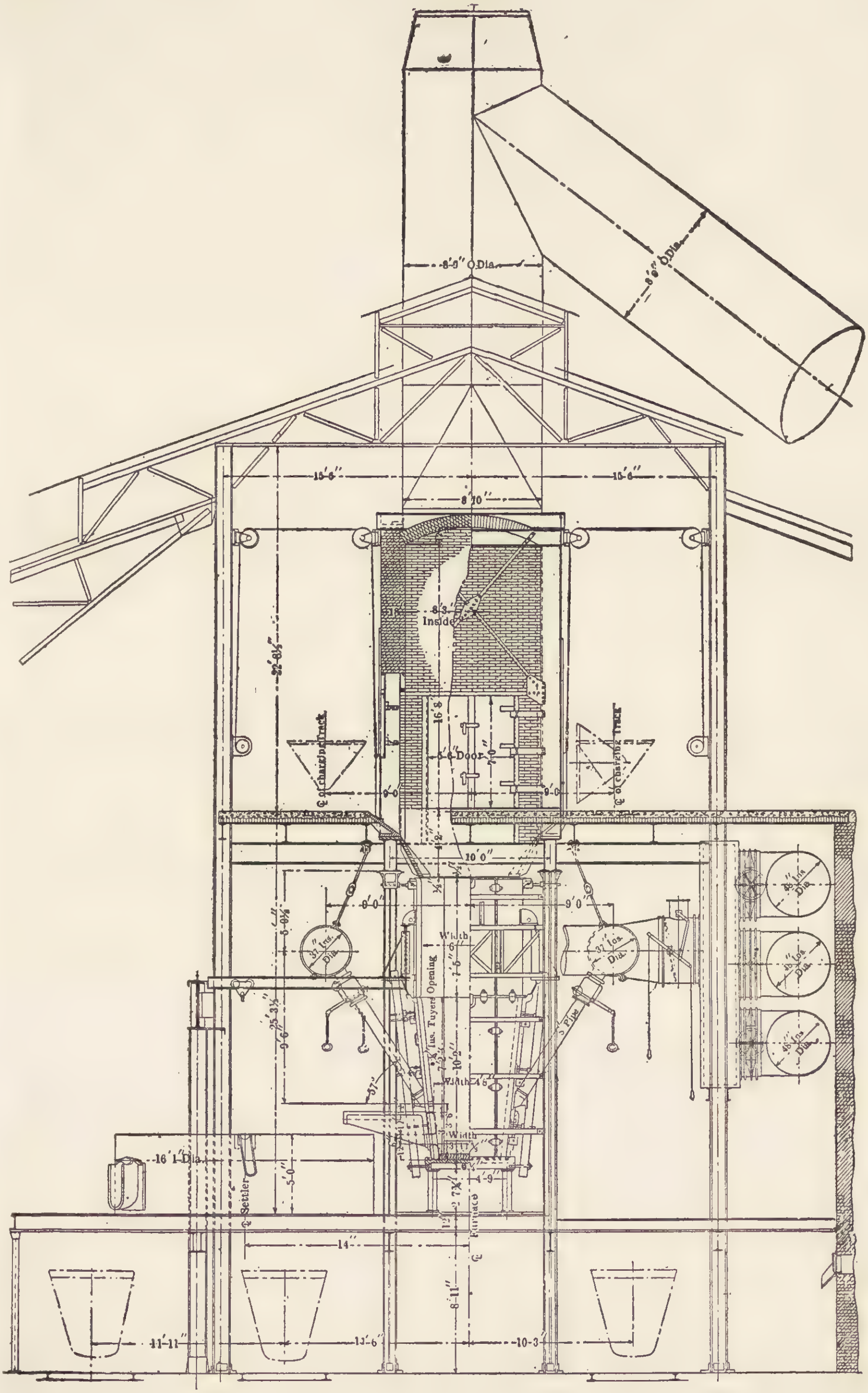

FIG. 23.—TENNESSEE COPPER COMPANY—TRANSVERSE SECTION OF 56 BY 270-IN BLAST FURNACE.

Herewith are given the results of an average week's run, in which two furnaces were smelting straight sulphide ores, and the third concentrating the first matte, and also putting through some custom-matte. The converter-slag is at present treated in the concentration furnace, and hence, when discharged, still contains 0.6 per cent. Cu.

When a furnace gets in bad condition through crusting, the practice is to run it down, adding several charges of coke and slag. The blast pressure is then reduced and the furnace cut down at the sides with bars 1.5 in. diameter having chisel ends of 2x4 in. steel. A chain, slippped over the top

TYPICAL CONCENTRATION RUNS

	Dec. 9 to 15 1904. Tons.	Dec. 16 to 22 1904. Tons.	Jan. 2 to 8 1905. Tons.
Sulphide ore	......	....	279
Converter slag	130	210	162
Blast-furnace slag	......	....	320
Quartz flux	562	527	307
Custom matte, 45%	......	....	252
First matte, 10% to 12%	1,296	1,285	808
Total	1,988	2,022	2.118
	Tons.	Tons.	Tons.
Coke used	89	71	151
Coke percentage	4.5	3.5	7.1
Slag Analyses.			
Copper	1.01%	1.03%	0.65%
Iron oxide	58.6	58.4	54.2
Silica	33.9	34.2	38.8
Matte Produced.			
Copper	48.2	52.2	44.2

of the bar, is attached to one of the electric locomotives, which slowly pulls it and the crusts over into the furnace—a great improvement on the older way of attacking the crusts with smaller bars and by hand alone.

Cananea Smelting Plant[13].—At the works of the Cananea Consolidated Copper Company (Greene Consolidated), at Cananea, Sonora, Mexico, the ore is smelted to a matte, which is then bessemerized to blister copper. There are eight blast furnaces, varying from 42x120 in. to 56x180 in. at the tuyeres, with a smelting column of approximately 11 ft. They have forehearths, that of the large one being of 15 ft. diameter, which, as well as the furnace crucible, is lined with chrome brick. The capacity of the furnace is 450 to 525 tons daily, with a blast pressure of 28 oz. Furnaces Nos. 2 to 7 have large closed tops, 20 ft. square and 22 ft. high. A sheet steel flue 400 ft. long, a steel and brick dust-chamber 61x180x20 ft., and a brick flue 17x20x400 ft. serve, in series, to connect the various furnaces to a stack of 20 ft. diameter having a hight of 200 feet.

The matte from the forehearths is drawn off into ladles (holding 67 cu. ft. each), which are handled by two electric overhead traveling cranes of 50 ft. span.

[13] *Mining and Scientific Press*, Nov. 25, 1905.

The cranes, of 40 and 50 tons respectively, are each provided with two 5-ton auxiliary hoists. The tracks are supported by 30-in. continuous plate girders carried by 8-in. Z-bar columns. The slag is removed both in cast steel and in cast iron 33 cu. ft. side-dumping slag bowls, mounted on trucks of the Colorado Iron Works type. These slag trucks are handled by five electric locomotives, two of them being of the single-motor type of 20 h. p., and three of the double-motor type of 40 h. p. per motor. The slag-dump to the east of the plant is 60 ft. high, and on the west the track runs through a tunnel to a gulch, which affords another dumping-ground.

The converter building is a steel-frame structure, roofed with heavy corrugated iron. It is 296 ft. long, 65 ft. wide, and has 37-ft. walls.

There are six stands for converters of the trough type, having hydraulic operating mechanism, and 22 converter shells each 8 ft. diameter by 11 ft. 6 in. length, and having 10-in. blast connections. The shells are of 1 in. steel plate and have 14 tuyeres 1⅛ in. diameter. The hydraulic cylinders are 24 in. diameter by 7 ft. 8 in. stroke, and give a revolution through 270 deg. The rack is thrown in and out of gear by means of a screw and a sliding cross-head operating a pair of toggle links. The water pressure for operating these cylinders is 200 ℔. per sq. in. and is taken from two accumulators, each 30 in. diameter by 10 ft. stroke.

The converter fumes are received by hoods and branch pipes moving on a steel track immediately above the stands. Through these the fumes pass to 48-in. uptakes and to dust chambers each 8 ft. 6 in. by 9 ft. by 14 ft. 6 in., these latter being carried on the roof of the building and branching to an 8-ft. flue, terminating in a plate-steel stack 8 ft. diameter by 125 ft. high.

The blister copper is cast from the converter direct into ingots of 300 ℔., the ingot molds being arranged in sets of six upon steel cars which run beneath the converter. The bars are trimmed and sampled by an electrically operated drill. The converter slag is poured into large molds 12 in. deep, is allowed to solidify, and is then broken up and returned to the blast furnaces.

Tacoma Smelting Works.—The output of these works in copper during the year 1904 was 16,125,744 ℔., valued at $2,418,866.10. The copper department consists of a blast furnace, converters and an electrolytic refining plant. In addition, copper-lead matte from the lead furnaces is treated. At the copper smelting plant, the ore, arriving either by water or by rail, is unloaded by electric hoist into hopper-bottom bins. From them it is carried by electric cars to the sampling mill, where it is coarse-crushed through a gyratory (Gates) crusher, sampled through a Vezin automatic sampler, and delivered to a 24-in. belt conveyor, 438 ft. long,

which raises it 52 ft. to a central distributing bin, from which it is spouted to the desired storage bins.

The capacity of the blast furnace is 300 tons per day of 24 hours, producing 60 tons of 50 per cent. matte. The matte is transferred by means of a 20-ton traveling crane from the forehearth of the furnace to the converter by means of 5-ton ladles, giving 32 tons daily of blister copper of 99 per cent. This is remelted into anodes of 99.25 per cent. Cu.

The refinery, 63x272 ft., has 250 tanks in series, each holding 16 to 20 anodes, or 4500 lb. The tanks, 8 ft. long, 2 ft. 6 in. wide and 3 ft. 6 in. deep, are of wood, lead-lined. The anodes are 26x35 in., 1 to $1\frac{1}{4}$ in. thick, weighing 250 to 310 lb. The cathodes, also 26x33 in., are starting plates, $\frac{1}{64}$ in. thick.

Two 300-kilowatt direct-current generators are used, driven by two 450 h. p. alternating current motors, and furnish a current of 3500 to 4000 amperes at 130 to 140 volts. The copper conductors are 3 in. square. The anodes are handled by a 65-ft. traveling crane—a tankful (18) together. The current density is 18 amperes per sq. ft., and the anodes are dissolved in 21 days to 10 per cent. of their original weight. Cathodes contain 99.99 per cent. Cu., and are remelted to be cast into plates, ingots and wire bars for local consumption.

Concrete Chimney.—The Tacoma Smelting Company, Tacoma, Wash., has built a chimney, 300 ft. high, of reinforced concrete. The base of the stack is on a hill 150 ft. high, making it 450 ft. above sea-level. The total hight of the stack is 306 ft. 6 in., the hight above grade being 300 ft. The foundation is 39 ft. 6 in. square. The inside diameter of the stack is 18 ft., the outside diameter just above the base 21 ft. To the hight of 90 ft. the walls are double, the thickness of the outer shell being 9 in., the air space 5 in., and the inner shell 4 in. The single shell of the stack is 7 in. The foundation is reinforced by crossing layers of steel bars, diagonal and parallel. The foundation bars are bent to terminate vertically in the stack, thus affording an efficient anchorage. The steel reinforcements of the chimney consist of vertical bars and horizontal rings, the vertical bars being proportioned to take all wind pressure. The rings are T-bars, $1x1x\frac{1}{8}$ in., spaced 3 ft. apart in the outer shell and 18 in. apart in the inner shell. This stack required two months to build, and was constructed by the Weber Steel Concrete Chimney Company, Chicago, Illinois.

American Copper Reverberatory Practice.[14]—In this series of articles Dr. Peters shows how, in the United States, it has been possible to increase furnace tonnage, and at the same time obtain clean slags. The principal aim is to get a quick melt due to high temperature. Now, in ordinary practice, only one-fourth of the time is ordinarily taken in melting down, the rest being for other operations,

[15]E. D. Peters, *Metallurgie*, 1905, pp. 9, 41, 63.

with the furnace open and cooling down to a dull red heat. These are fettling, charging, spreading, cleaning the surface of the hearth, grating, tapping and skimming both of slag and matte. Unless operations are conducted under the best conditions of draft, fuel and working, fusion is delayed.

Under ideal conditions the maximum output would be obtained:

(1) By having a furnace of the largest possible dimensions, so as to obtain easily a temperature of 1600 to 1700 deg. C., when the normal charge calls for 1400 deg.

(2) By having the operations so arranged as to shorten the time of cooling down.

Small furnaces do not give good results in these respects.

It is necessary, first of all, to burn coal rapidly. There are three factors to consider, viz., grate-area, draft and firing. Experience shows that the coal burned per hour per square foot of grate-area is invariably proportional to the ratio of cross-section of the stack to the grate-area. The quantity of charge melted is proportional to the coal burned per square foot of grate-area. In the United States, the best results have been obtained on grates burning 46 lb. of coal per square foot per hour. A sharp draft is needed,—a high coal consumption,—and it should be looked after and measured. Practice is often satisfied with a draft of 0.7 in. of water when 1.5 in. would give more rapid and economical fusion. It is customary in American practice to have a high common stack and a natural draft.

Now comes in the question of best utilization of heat—the dimensions of the hearth. The width cannot exceed 20 to 21 ft. As for the length, it is necessary to have the temperature of the gases high enough to readily melt the slag. Former (though recent) Montana furnaces were 50 ft. long, using 37 down to 27 per cent. of coal. The present furnaces are 102 to 117 ft. long, requiring only 21 per cent. coal.

As regards firing, we may assume, under ordinary practice, the coal to be moderately long-flaming, with 10 per cent. ash. Grating, however promptly performed, cools the furnace so that it takes an hour to recover, and this has to be done two or three times in 24 hours. After each firing the temperature falls for five minutes at least, and it is ten minutes before the heat is up again. In the recent furnaces, the temperature is high enough, and the mass of coal so great that the dropping of a charge of 3000 lb. of coal makes but an insignificant difference in the appearance of the flame of the hearth. At the end of three or four minutes it has become bluish white and transparent again.

In furnaces with forced draft the grates are cleaned at the time a charge is dropped. When the coal ash produces fusible clinkers, they are made brittle by wetting the clinker-bed with a hose.

As regards the loss of heat by radiation of the furnace, improvement has

been made by the use of silica brick. For the roof, 12-in. brick are used in place of the 9 in., as heretofore. The foundation is made solid with a monolith of slag taken from adjoining furnaces, and on this comes the sand bottom. The flues and channels, which used to be prepared in the brickwork of the furnace for pre-heating the air, have been done away with, as well as the arch beneath the hearth.

Dr. Peters draws attention to the fact that a saving of heat and of time is made by charging the calcines hot from the roasters. The charging is done in the first 20 ft. of the furnace near the fire-bridge. The hot ore spreads out to the borders of the furnace. Since a large body of matte is kept on the hearth, the newly dropped charge speedily gets heat, both from the flame above and from the highly heated matte below.

It is a great help to charge calcines hot and to mix them with their own weight of silicious ore in the roaster, by which in a specified furnace the output has been raised from 32 up to 39 tons daily.

The careless opening of the doors contributes much to the cooling effect, except the skimming door at the front end, which is little harmful, because of its position near the uptake. The tapping should be done as quietly and regularly as possible. Crushed quartz is best suited for fettling; this need be done only once in four to six weeks.

The high temperature which it is thus possible to maintain permits the use of slags with 40 to 47 per cent. silica, which do not attack the brickwork to the same degree that the more basic ones, needed at the lower temperature, did.

Following are some of the particulars of the large furnaces: Fire-box, 7x16 ft. or 112 sq. ft. area. Hearth, 20x103 ft. Ratio of grate to hearth area, 1:16. Coal consumption per 24 hours 58 tons, or 21 per cent. of the charge. Capacity of the furnace, 275 to 330 tons per day. By concentrating the droppings from the grates, there is recovered six tons of coke daily. The escaping heat from the furnace is utilized in heating two steam boilers by which 600 h. p. is obtained. Every 80 minutes a charge of 14 tons is dropped upon the matte bath, which amounts to 100 to 150 tons, and is of the depth of 6 or 8 in. There is tapped off into converter ladles 9 or 10 tons of matte at a time, and every three or four hours 30 to 35 tons of slag, which is run out in about 15 minutes. It is granulated and sluiced away to the dump.

Copper Smelting at Kedabeg, Russia.[15]—Petroleum is largely used for fuel at these works. It has a specific gravity of 0.882, and contains 87.4 per cent. carbon, 12.5 per cent. hydrogen and 0.1 per cent. oxygen, and has a heating value of 11,700 calories. It costs, delivered, 16c. per gal. In practice it may be assumed that 54 parts of petroleum has a heating value equivalent to 100 parts of coal or 250 parts of wood. Notwithstanding

[15] *Trans.* Institution Mining and Metallurgy, April 13, 1905.

its high price, petroleum is still the cheapest fuel which can be used at these works, where it is employed for roasting, smelting and refining, for steam-making and for burning lime and brick.

The Kedabeg ores are sulphides, and also very earthy, and contain much blende and barite. The pyrite ores form the chief bulk, with fines up to 60 per cent. Ores carrying over 5 per cent. Cu are smelted direct; the low grade sulphides of 2 per cent. Cu are sold to the Baku kerosene works, for the manufacture of sulphuric acid. The lump ore is roasted in kilns to 8 or 10 per cent. S; that of 0.5 to 0.25 in. in size is roasted in

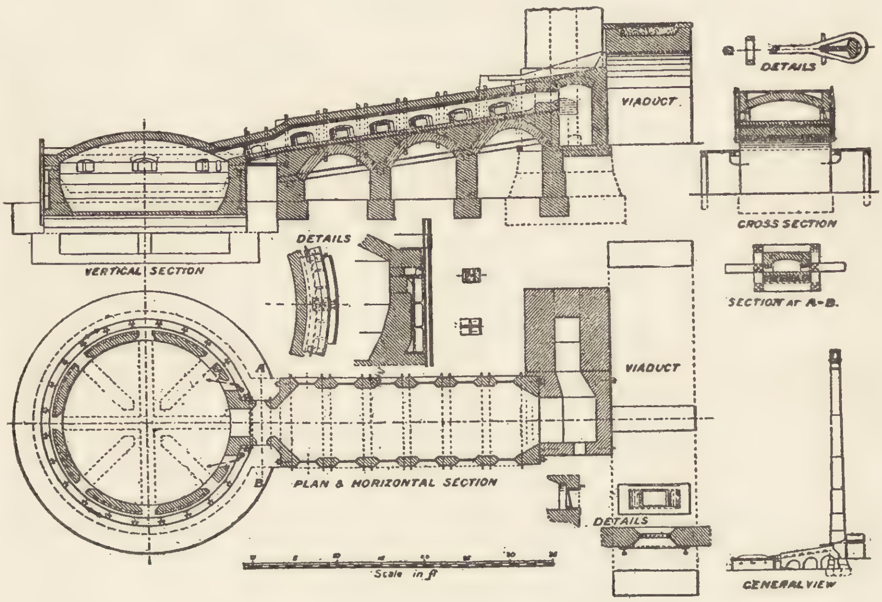

Fig. 24.—Reverberatory Furnace.

reverberatories, and all that is smaller than 0.25 in. in Gerstenhöfer kilns to 6 per cent. S, or, when fired with petroleum, to 3 per cent. S. In this kiln two men can roast 25,200 lb. in 24 hours. About 1 per cent. flue dust is made.

The reverberatory smelting furnaces are connected to roasting hearths, as shown in Fig. 24, and the roasting is performed by the waste heat from the smelting hearth. Each furnace roasts 11 to 18 (average 14) tons in 24 hours. There are two men per shift of 12 hours, each paid 21c. per shift. There are six smelting-reverberatories, circular in plan, 29 ft. in diameter. Each furnace is of 36 tons capacity, and has a high arched roof (of 32 in. rise) and a sand hearth. The borders of the hearth are fettled with finely screened dead-roasted matte. The furnace is fired by jets of petroleum,

one on either side of the outlet flue, the oil entering under a head of 30 ft., and in a tangential direction, as shown on the drawing. The two flames sweep round the circumference of the furnace, returning at the middle to the flue, where the waste gases escape into the roasting hearth, and thence to the stack, 100 ft. high. The smelting furnace is charged at the five doors there provided. The charge at the roaster is withdrawn at its

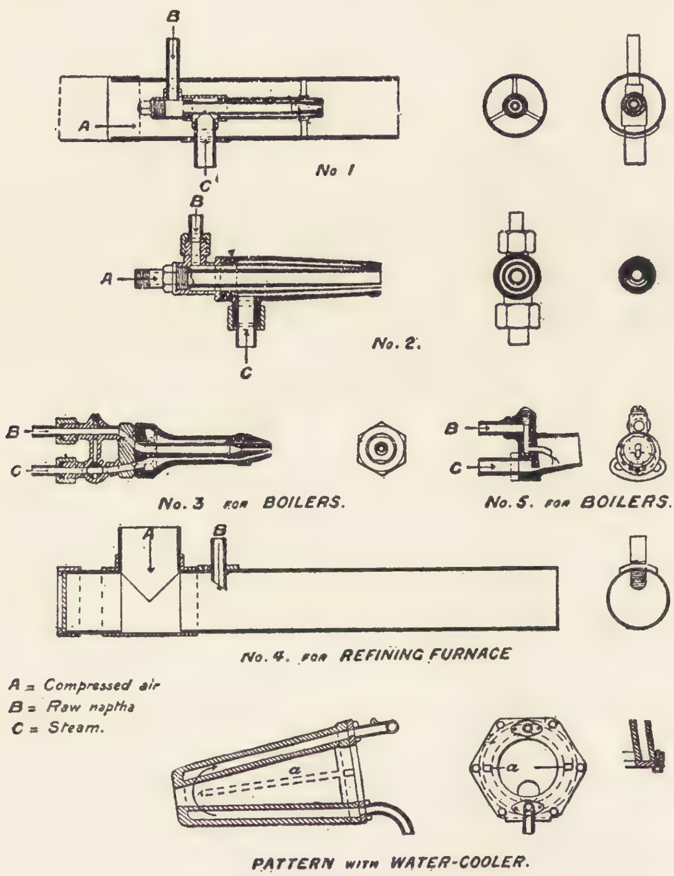

FIG 25.—OIL BURNERS

last doors and is recharged into the smelting furnace. The blende (associated with the barytes) in these ores is very troublesome, the zinc compound separating with difficulty, and partly entering the matte. Because of these troubles, there should be more than 6 per cent. S left in the charge, since otherwise the reactions are irregular and infusible masses separate out on the hearth.

A recent analysis of the matte showed 28.6 per cent. Cu; 36.7 per cent. Fe;

4.1 per cent. Zn; 0.8 per cent. N and Co.; 0.1 per cent. PO; 1.0 per cent. SiO_2; 0.1 per cent. Al_2O3; 8.6 per cent. $BaSO_4$; 19.7 per cent. S; 0.05 per cent. As (Sn); 7.3 oz. Ag and 0.4 oz. Au per ton.

The high proportion of iron in the ores gives a basic slag of the following composition, viz.: 23.9 per cent. SiO_2; 50.5 per cent. FeO; 2.5 per cent. ZnO; 0.3 per cent. Al_2O3; 0.8 per cent. MgO; 4.0 per cent. CaO; 16.1 per cent. $BaSO_4$; 0.6 per cent. Cu_2O; 0.1 per cent. As (Sn); 1.1 per cent. S.

The kinds of burners used are shown in Fig. 25. The burner No. 1 has proved the best for ore smelting, the air pressure being 1 to 1.2 in. mercury,

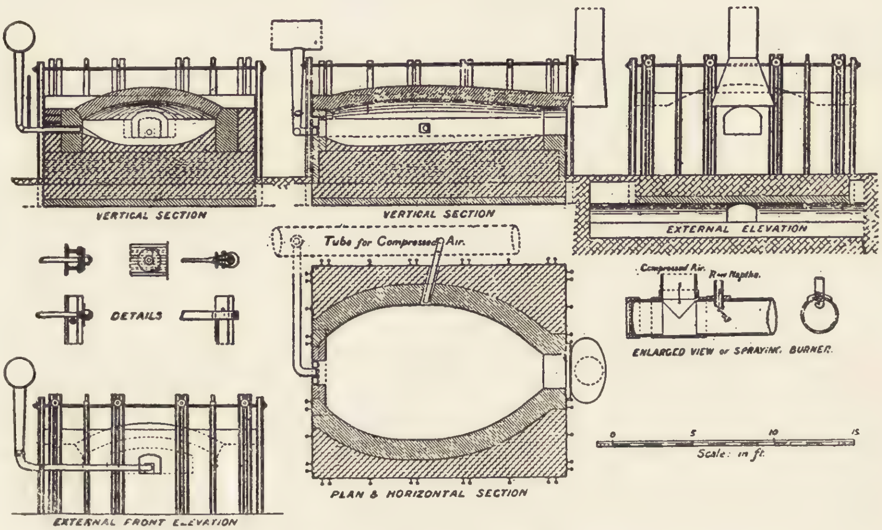

FIG. 26. REFINING FURNACE.

and the steam pressure 60 to 90 lb. per sq. in. With these burners a temperature of 1500 deg. C. has been attained.

If blast at high pressure is available, burner No. 2 gives excellent results. The simple plan of admitting the oil under high pressure, without using steam, has, however, never proved successful.

Each charge for the smelting furnace consists of 2160 lb. roasted lump ore, 1080 lb. roasted middlings from the calciners, 2160 lb. roasted fine ore from the Gerstenhöfer kilns, and 530 to 720 lb. of silicious ore of 50 per cent. SiO_2. The hearth will hold 72 to 108 tons of material. When the charge has been melted down to quiet fusion, the slag is removed at the door opposite the flue. The matte is tapped off, as necessary, into sand beds. The charges and materials about the works are handled in self-tipping cars of two to three tons capacity. Each furnace needs four men per shift of 12 hours, with one foreman to three furnaces. The men are

paid 14.2c. per ton of ore smelted. The oil used at the smelting furnace amounts to 18 per cent. of the charge, or, allowing for delays, to 22 per cent. of the roasted ore charged. The matte produced from these smelting furnaces is roasted in heaps or in stalls, and is then run in blast furnaces to produce black copper ready for the process of refining.

Fig. 26 gives the form and arrangement of the refining furnaces of a minimum capacity of 7.2 tons in 24 hours, or of four charges of 3600 to 5400 lb. black copper. The oil-jet is sprayed into the furnace at the middle of its long side, and is met in addition by an air-blast at the working-door end, which serves to complete the combustion of the gasified petroleum. The flame thus sweeps over the hearth and escapes by means of a short stack of sheet iron. The time of melting down is performed in 2 to 2.5 hours, with the blast at low pressure, which effects a complete combustion of the oil, and partly oxidizes impurities. The blast is then turned on the molten bath, at the full pressure of 1 in. of mercury, through a tuyere directed downward upon the surface of the metal. Sulphur, arsenic and antimony are volatilized, much smoke being given off. The zinc and iron begin to slag off, together with some copper oxide. The slag is skimmed as it forms, this operation lasting 1 to 1.5 hours, after which the rabbling, or roasting, begins. This latter operation is characterized by the escape of bubbles of sulphurous-acid gas at the surface of the bath, due to the well-known reaction between the oxides just formed and sulphide of copper of the black copper. This takes 2.5 to 3 hours.

When the surface of the bath has become perfectly quiescent the poling begins, which is continued until a sample withdrawn with a spoon shows the characteristic cubic structure, quite free from blow-holes.

After this the final reduction of the copper oxide is proceeded with, poling being completed under a covering of coal dust until the copper reaches the desired degree of strength and toughness, characterized by a rose-colored fracture and silky structure, and by standing hammering and bending without cracking, both hot and cold. The amount of oil equals 75 per cent. by weight of the copper produced. For two furnaces there are needed per shift one foreman, one furnaceman, and four helpers, these latter for charging and tapping.

Kedabeg copper has the following composition: 99.57 per cent. Cu; 0.027 per cent. Pb; 0.038 per cent. As; 0.06 per cent. Sb; 0.031 per cent Ni and CO; 0.009 per cent. Fe; trace Bi; trace S; 7.8 oz. Au and Ag per ton.

Constitution of Mattes Produced in Copper Smelting.[16]—Mattes, produced in copper smelting, range generally from 30 to 80 per cent. in copper, but matte of 50 per cent. Cu is found to be the most desirable for shipping or for converting. Such mattes are composed of copper, iron and sulphur, with other elements, usually in proportions sufficiently small

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[16]Trans. American Institute of Mining Engineers, *Bi-Monthly Bulletin*, Nov. 1, 1905.

to allow of their treatment as impurities rather than as essential constituents. In the modern practice of the smelting of copper ores in blast furnaces, the tendency is to approximate to pyritic smelting, the furnaces being low and the air blast in excess; hence the furnace atmosphere is neutral or oxidizing, and the iron of the charge is never reduced beyond the ferrous form.

These mattes receive names as follows:

(1) A matte of 35 to 55 per cent. Cu, hard, compact, and of a dull-bronze color, and containing no visible copper. This is known as "coarse" metal.

(2) A matte of from 60 to 70 per cent. Cu of a bluish purple color, moss copper separating out. This is known as "blue metal."

(3) Matte of 70 to 76 per cent. Cu, having a white unevenly plated appearance. It is called "white metal." Though apparently homogeneous, it frequently contains some visible metallic copper.

(4) A matte of 78 to 81 per cent. Cu receiving various names, as "purple metal," close regule and spongy regule, according to its appearance. Metallic copper is visible in considerable proportion.

To determine the chemical combinations in which copper, iron and sulphur occur in mattes, a neutral solution of silver nitrate was used as a reagent, acting as follows: It rapidly decomposes cuprous sulphide (Cu_2S), its silver is precipitated both by metallic copper and by cuprous oxide (in the latter case with the formations of an insoluble basic cuprous nitrate), and very slowly by metallic iron; but it has no action on ferrous sulphide (FeS).

Experiments were instituted to determine what compounds of copper, iron and sulphur were stable at the temperature of fusion of matte. (1) It was found that cuprous sulphide (Cu_2S) was a stable compound, and that cupric sulphide was converted into it in presence of metallic copper. (2) Ferrous sulphide is a stable compound, and it is capable, when fused, of taking up a considerable proportion of metallic iron. (3) Metallic iron decomposes cuprous sulphide with the formation of iron sulphide and metallic copper; i. e., sulphur has a greater affinity for metallic iron than for copper. Ferrous copper dissolves in metallic copper in all proportions up to 120 per cent. of its own weight. (5) Cuprous and ferrous sulphides may be fused together in all proportions, forming an apparently homogeneous mass. (6) Most of these double sulphides have the property of dissolving metallic copper; however, a high-grade matte of approximately 90 per cent. Cu_2S and 10 per cent. FeS, having all the characteristics of white metal, and of the composition ($5Cu_2S,FeS$) appears to form a dividing line between these sulphides and those higher in copper, where such copper, as in purple metal, is held in mechanical suspension. We may conclude, therefore, that all grades of matte up to white metal consist of white metal united to an excess of ferrous sulphide, and this

excess is the agent which dissolves the metallic copper occurring in these mattes.

Pyrometric Measurements.—The following table shows the melting points of mattes of the percentage compositions indicated, from which we gather that the melting point of the white metal is higher than that of any other matte and even above the melting point of copper;

TABLE I.

Sample	Copper Content Per Cent.	Melting Point Degrees C.
A	32.6	875
1	49.7	955
3	61.2	1070
6	71.7	1121
11	80.1	1098
Metallic Copper	100.0	1083

The existence of the compound 5 Cu_2S,FeS in mattes puts us in position to explain the phenomena attending their concentration in copper smelting. The principal chemical reactions which take place result in the formation, at all stages, both of cuprous sulphide and of metallic copper, the latter being held in solution in the ferrous sulphide or in the higher grades mechanically admixed. Under the conditions of formation, ferrous sulphide becomes, when molten, super-saturated with metallic copper, which it dissolves to the extent of 60 per cent. of its own weight; upon cooling, a portion of the copper separates out with the formation of the moss copper, with which we are familar in the case of blue metal.

When the concentration is carried beyond the stage of white metal, the mattes contain metallic copper, but in mechanical suspension only. The principal action at this stage appears to be the decomposition of the compound 5 Cu_2S,FeS. As this compound is broken up with the formation of cuprous oxide, the latter, reacting on the white metal, precipitates copper in metallic form as "bottoms."

Table II gives examples of the constituents of two mattes containing the ordinary proportions of impurities. Such impurities are in general sufficiently small in quantity to make immaterial the exact state of chemical combination in which they occur.

TABLE II.

Components	1	2
	Per Cent.	Per Cent.
Copper......	72.91	54.06
Iron........	5.00	22.15
Sulphur....	21.15	22.50
Arsenic....	0.47	0.23
Antimony...	0.11	0.04
Bismuth	0.06	0.06
Nickel	0.25	0.87
Total	99.95	99.91

From the readiness with which arsenides and antimonides are formed, as well as the instability of the sulphides of these elements, we may assume

that they replace sulphur in the mattes, combining with iron. Other common elements occur as sulphides.

TABLE III.

Rearrangement of Data in Table II in Order to Show the Constitution of the Materials.

Components.	1	2
	Per cent.	Per cent.
Cuprous sulphide	91.16	45.20
Metallic copper	0.20	18.25
Iron and nickel sulphides	7.58	35.99
Bismuth sulphide	0.07	0.07
Iron and nickel arsenide	0.82	0.35
Iron and nickel antimonide	0.16	0.06
Total	99.99	99.92

Assuming the presence of white metal, 5 Cu_2S,FeS, we have:

TABLE IV.

	1	2
	Per cent	Per cent.
White metal	83.3	50.2
Cuprous sulphide	14.3	*Nil.*
Ferrous sulphide	*Nil.*	31.0
Metallic copper	0.2	18.2
Total	97.8	99.4

Impurities in mattes have a tendency to reduce the proportion of white metal (5 Cu_2S,FeS). Thus a 75 per cent. Cu matte from impure ores will contain less than 2 per cent. Fe, while a pure matte of 78 per cent. copper will still have full 2 per cent., and it is only when much copper has been separated out that the matte is freed from iron. Thus the purer mattes of over 70 per cent. are concentrated with greater difficulty than the impure ones. A 72 per cent. impure copper matte will contain 14 per cent. Cu_2S and 83 per cent. of the white metal, whereas a pure matte of that grade contains only 2 per cent. Cu_2S and 97 per cent. white metal. It appears that all the white metal must be decomposed before much metallic copper can form, and as impurities, as shown, decrease the proportion of white metal, their presence tends to facilitate the separation of copper. Actual practice indicates the same effect.

Conclusions.—The following conclusions may be deduced:

(1) Cuprous and ferrous sulphides will combine to form a chemical combination, 5 Cu_2S,FeS, already named "white metal."

(2) White metal enters into the composition of all mattes.

(3) In fusions white metal mixes in all proportions with ferrous sulphide, these constitutents separating on cooling.

(4) In all the lower-grade mattes below 70 to 76 per cent. Cu, the excess of ferrous sulphide is capable of dissolving metallic copper, part of which, at the stage of blue metal, separates out as moss copper.

(5) In the fusion of mattes above the stage of white metal, that compound and cuprous sulphide (Cu_2S) may mix in any proportion, but it is only in mattes where the white metal is in small proportions that copper is readily separated by processes of oxidation.

(6) Fused mixtures of white metal and cuprous sulphides hold copper and cuprous oxide in mechanical suspension.

(7) Impurities, by displacing a part of the ferrous sulphide available for the formation of white metal, reduce the proportions of the latter compound.

Miscellaneous.

Recovery of Copper from Mine-Water.[17]—The water of Silver Bow Creek, near Butte, Montana, contains appreciable quantities of copper in solution, resulting from the drainage of adjoining mines and reduction works. For this, a tower 20 ft. high, 20 ft. long and 8 ft. wide has been erected, which carries seven lattice shelves covered with old sheet-iron and tinned iron scrap. An electric pump at the creek supplies 1000 gal. of water per minute, which is distributed by troughs over the tower. The water flows through holes bored in the bottom of the troughs to ensure even distribution. Finally two wide troughs at the bottom of the tower, filled with iron scrap, carry the water away. While the water thrown by the pump is green in color, no evidence of copper can be observed in the escaping water. Several tons of precipitate have been realized from a single clean-up.

Manufacture of Copper Tubes by the Elmore Process.[18]—The Société Anonyme Elmore-Schladern-Sur-Sieg., Germany, uses this process. The lead-lined tanks are 20 ft. long by 6 ft. 8 in. wide. The rotating mandrels are supported by glass bearings, and the agate burnisher travels over the surfaces at a rate such that, between two successive passages, the thickness of the depositing surface has increased by 0.06 in. to 0.07 in. The copper-sulphate electrolyte carries 3 per cent. of free sulphuric acid. The current necessary for 40 tanks is 800 amperes at 50 volts, and the current density is 18 amperes per square foot. The generator is driven by a 60 h.p. engine. The production monthly per horse-power is 220 lb. The slimes, deposited on the bottom of the tanks, often contain gold and silver. These works turn out 1500 tons of pipe per annum, some of them being as much as 100 in. in diameter, 20 ft. long and weighing 3 tons each.

Elmore tubes, as taken from the tank, show a strength of 3725 lb. per sq. in., with an elongation of 16.1 per cent. When cold drawn they have a strength of 34,839 lb. per sq. in., with an elongation of 27.8 per cent. When annealed at 500 deg. C. the rough tubes have a strength of 30,288 lb., but with an elongation of 49.6 per cent. When cold drawn and then annealed, the strength drops to 30,000 lb. with an elongation of 50 per cent. Compared with a cold-rolled-drawn annealed tube we find this latter to have a strength of 31,710 lb., with an elongation of 47.9 per cent.

[17] *Engineering and Mining Journal*, LXXIX, 966.
[18] *Behang till Jernkortereis Annaler*, 1904.

COPPERAS.

The amount of copperas made and marketed in the United States during 1905 was 21,103 short tons, valued at $147,721, as against 16,956 tons (worth $118,692) in 1904.

The principal supplies of this substance come from the iron and steel sheet and wire plants, where sulphuric acid baths are employed to clean the rolled or drawn product preparatory to tinning or galvanizing. The United States Steel Corporation, through its subsidiaries, the American Steel and Wire Company and the American Tin Plate Company, is thus the heaviest contributor to the country's output, followed by the Pennsylvania Salt Manufacturing Company.

PRODUCTION OF COPPERAS IN THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Sh. Tons	Value.	Year.	Sh. Tons	Value.	Year.	Sh. Tons	Value.
1894	14,877	$104,100	1898	11,285	$58,105	1902	19,784	$118,474
1895	14,118	69,846	1899	13,770	108,508	1903	20,240	121,440
1896	11,170	52,662	1900	12,374	96,517	1904	16,956	118,692
1897	11,924	56,565	1901	23,586	112,336	1905	21,103	147,721

A considerable portion of the copperas made by the chemical manufacturers, notably C. K. Williams & Co., Allentown, Pa., the S. P. Wetherill Company, Philadelphia, and the Stauffer Chemical Company, San Francisco, is converted at once, by roasting, into Venetian red, for the paint makers; none of the production so used, at first hand, is included in the above totals. The dynamite manufacturers maintain a small but steady output of copperas, recovered in their processes.

CORUNDUM AND EMERY.

By Edward K. Judd.

Domestic production forms only an insignificant part of the consumption of corundum and emery in the United States, as the following table, in which the two abrasives are not separately treated, indicates:

STATISTICS OF CORUNDUM AND EMERY IN THE UNITED STATES.

Year.	Production. (a)		Imports. Grains.		Imports. Ore and Rock.		Other Mfs
	Short Tons	Value. (b)	Pounds.	Value.	Long Tons.	Value.	Value.
1896	2,100	$106,000	751,464	$26,520	6,289	$119,667	$1,971
1897	2,193	111,810	520,095	20,022	5,209	107,644	2,211
1898	3,742	207,430	577,655	23,320	5,547	106,269	3,810
1899	3,970	228,570	728,229	29,124	7,435	116,493	11,514
1900	5,030	247,100	661,482	26,520	11,392	202,980	10,006
1901	4,305	146,040	1,086,729	43,217	12,441	240,856	10,926
1902	4,251	104,605	1,665,737	49,107	7,157	151,959	13,776
1903	2,542	64,102	3,595,239	109,272	10,884	194,468	17,829
1904	1,932	57,235	2,281,193	109,772	7,054	138,931	11,721
1905	(c) 2,315	19,677	3,209,914	143,729	11,072	185,689	17,996

(a) Statistics of the United States Geological Survey for 1901 and subsequent years, except 1905. (b) Values have not much significance owing to the wide variation in the quality of the materials combined in the totals. (c) Emery only.

The production of these natural abrasives is being curtailed by the expanding use of the artificial products carborundum, alundum and crushed steel, with which they come into direct competition. Garnet and quartz, on the other hand, not entering into the same fields of usefulness, do not show a similar curtailment.

Corundum is the crystallized oxide of aluminum, Al_2O_3, but contains usually small proportions (less than 1 per cent.) of iron, silicon and water. It occurs as masses and as hexagonal crystals in the more basic igneous rocks or their metamorphic derivatives. Its commercial value lies in its hardness (9) and in its irregular fracture.

Emery is an indefinite mixture of corundum, magnetite or hematite, spinel, garnet, staurolite and other hard minerals. Its value as an abrasive depends altogether upon its proportion of corundum, for which reason the products from different localities show wide variations in abrasive quality. The percentage of alumina in an emery ore is not an indication of its abrasiveness, since spinel, garnet and staurolite all contain alumina, and the quantity of these minerals, of inferior hardness, does in some ores exceed the quantity of corundum. It is now conceded that the only safe test of an emery is to form it into a grinding tool, in the usual way, and to apply the tool to regular work.

Corundum Mining in the United States.

North Carolina.[1]—Corundum is found in association with peridotite in 15 counties in North Carolina, mining for this mineral having begun here in 1871 at the Corundum Hill mine, now owned by the International Emery and Corundum Company. The Buck Creek mine, in Clay county, owned by the same company, if it were more accessible by railroad, would, it is thought, prove one of the best corundum properties in the State. Other mines of interest are the Herbert, owned by the North Carolina Corundum Company, with a complete mill and railroad connection, and the Bad Creek mine, owned by the Toxaway Company, Sapphire, Jackson county.

Other Southern States.—There are scattering deposits of corundum in 13 counties of Georgia. The Laurel Creek and Track Rock mines are the only ones that have produced any considerable quantity of the ore. Corundum is found in four counties in South Carolina, and mining operations have been carried on in two portions of a belt in York county.

Montana.—The corundum deposits of Montana, in Gallatin county, are being developed by two companies, of which the larger producer is the Montana Corundum Company, which has a property well equipped with machinery for mining, cleaning and preparing the ore for market. Its cleaned corundum makes a satisfactory abrasive, and the ore can also be used in manufacturing vitrified wheels. The Bozeman Corundum Company is developing a mine 14 miles from Bozeman, and is making a regular output.

Corundum in Foreign Countries.

Canada.—The grain corundum produced from Raglan and Carlow townships, in the counties of Renfrew and Hastings respectively, during 1905, was 1644 tons, valued at $149,153, as against 1665 tons, worth $150,645, in 1904. The producing companies are the Canada Corundum Company and the Ashland Emery and Corundum Company (formerly the Ontario Corundum Company). The former plant is at Craigmont, Raglan township, and the latter at New Carlow, in the same township. Corundum Refiners, Ltd., has not yet begun production.

Statistics of corundum since its production here, beginning with 1900, are as follows:

CORUNDUM STATISTICS 1900 TO 1904.

Schedule.	1900	1901	1902	1903	1904	1905
Corundum produced.....tons	60	534	1,137	1,119	1,665	1,644
Value of product.....$	6,000	53,115	83,871	87,600	150,645	149,153
Workmen.....No.	35	68	95	186	202	....
Wages paid.....$	10,000	30,406	34,674	106,332	139,548	....

The Canada Corundum Company employs about 140 men, the mill

[1] J. H. Pratt, ***Bulletin*** No. 269, United States Geological Survey.

now concentrating nearly 200 tons of ore per day, with a corresponding output of 10 to 12 tons of corundum. Various new buildings have been erected, including an analytical laboratory.

The Ontario Corundum Company's new mill was burned early in 1904. Rebuilding was begun at once, with various changes suggested by experience. The two main buildings of the new plant are the boiler house and mill. In the former a 125-h.p. boiler will, besides running the plant, do all the drying by steam. The plant includes five Blake crushers—one 9x15 in., two 7x10 in., and two 4x10 in.; two impact crushers or pulverizers; two rolls; dividers; a Noble magnetic separator; seven Hooper pneumatic jigs; a 75-h.p. horizontal engine and an electric lighting plant. The ore will be dried at once on arrival from the mine, and will remain dry. The present mining force is 20 men.

India.—Corundum suitable for abrasive purposes is reported as occurring (though in small quantity) in the Pararapatti area, Salem district, upper Burma, the extension of corundum-bearing rock being about 24 miles. Another similar deposit is reported about 250 miles north of Calcutta. It is very probable that nearly all of the corundum deposits known in India are secondary minerals and the result of metamorphism.

Methods of Mining and Cleaning Corundum.[1]

Nearly all of the peridotite formations in which the corundum deposits occur are bold outcrops on mountain sides and hilltops, having almost perfect natural drainage. Though prospecting work is usually done by means of open cuts supplemented by tunnels, it is found inadvisable to break the surface more than is absolutely necessary, because if water or frost penetrates among the slippery alteration products, large masses are apt to fall, closing tunnels and shafts. Mining is therefore confined to a system of tunnels and shafts, preferably kept well timbered and drained. A large amount of the material to be handled is easily worked out with pick and shovel. Where, however, the corundum is found associated with feldspar, blasting is necessary.

If the mill is located some distance from the mine, and if the topography permits, a line of sluice-boxes can be built between the two, and the more or less finely divided ore may be carried to the mill in a water-current by means of these boxes. In this way the ore is partly cleaned by the time it reaches the mill.

To clean corundum, the ore is crushed and sieved; all that will not pass through a No. 12 screen is recrushed and rolled until the required size is reached. The unattached impurities are then removed by placing the crushed ore in boxes through which water flows in a stream. The product next passes through a screw or scouring machine, which grinds

[1] *Op. Cit.* Refer also to 23d *Annual Report* of the New York State Geologist, p. 167.

out almost all the impurities. The final traces are removed by a muller or "chaser" which causes the grains to rub together. The product is afterward dried by exposure to furnace or steam heat, and finally screened to the various sizes.

Emery Mining in the United States.

Only two localities in the United States now afford any output of emery: Chester, Mass., and Peekskill, N. Y. At the former, mining has nearly come to a standstill, the Ashland Emery and Corundum Company (successor to the Hampden company which operated the mines for so many years) keeping up a small production by picking over the dumps. A complete description of the emery industry at this place will be found on a following page.

New York.—The emery deposits in New York occur associated with the norite rocks of Westchester county. These deposits vary in character, some being worked for iron ore, others containing magnetite and corundum. Associated with most of the corundum is spinel; much of the ore that has been mined for emery has contained little or no corundum, but only magnetite and spinel. An ore of this sort has most of the requisite properties of a true emery except the high degree of hardness due to the corundum, so that when made into a wheel it would not have the cutting efficiency of a true emery wheel. For many purposes, however, wheels made from this material would be fully as satisfactory as carborundum or emery wheels. The Blue Corundum Mining Company, of Boston, is one of the largest miners in this section, operating three miles from Peekskill. The ore is shipped to Easton, Penn., for treatment. The Tanite Company, of Stroudsburg, Penn., is operating near here, as is also H. M. Quinn, of Philadelphia.

Southern States.—During the last few years emery has been found in Macon county, N. C., in what appears to be considerable quantity. Nothing but prospecting has, however, been done here as yet.

Emery has also been found in a number of localities in Pittsylvania county, Va., but no mining has yet been attempted.

Emery in Foreign Countries.[3]

Turkey.—Turkish emery comes from the province of Aidin, in Asia Minor, which embraces nearly the entire basins of the rivers Sarabat and Meander. Of this district Smyrna is the trade center, and the emery deposits are mainly within a radius of 125 miles. The occurrences of the emery in the different localities where it is found are all quite similar, it being imbedded in a bluish, coarse-grained to compact marble or limestone resting upon mica-slates, schists and gneisses. It occurs in pockets,

[3]*Op. Cit.*

scattered irregularly through the rock, that are sometimes 200 ft. in length and 300 in width.

Greece.—Considerable quantities of emery have been found at different points in the Grecian archipelago, the most important source being the island of Naxos, where the emery is found in large blocks, more or less mixed with the red soil or imbedded in marble. The islands of Apperonthos, Nicaria and Samos also contain deposits. The shipping point of all these localities is Syra, and as the emery ore is usually taken merely as ballast, its shipping cost is low and its value at foreign ports is only about 1c. per lb. This is one reason why there is such close competition bewteen this product and that from the United States.

The Emery Industry at Chester, Mass.

By Claude T. Rice.

The emery industry at Chester, Mass., has become one of milling rather than of mining, for, of the three emery companies having headquarters here, only one, the Ashland Emery and Corundum Company, owns any deposits of ore. The other two companies, viz., the Abrasive Mining and Milling Company and the Hamilton Emery and Corundum Company simply mill ore imported from Greece and Turkey. Heretofore, the Ashland company, successor of the old Hampden Emery and Corundum Company, has mined ore in some part of its mines, but within a year it has only picked over its dumps, in this way producing about 70 tons per month of ore carrying around 50 per cent. emery.

This suspension of mining is due mainly to the fact that no development work was done for several years when the old Hampden company was mining in the large pocket of ore that made Chester emery famous. Development work was resumed only when the pocket began to pinch out. Unfortunately the pockets of ore opened up in late years have not been large, and these have been exhausted faster than new ones have been found. Still the Ashland company is said to have quite a little corundum in the Wright mine, 2½ miles southwest of Chester.

The emery vein at Chester has a general northeast-southwest direction and dips about 70 deg. to the west. Granite forms the country rock in the vicinity, but the foot-wall of the vein is a chloritic rock locally called soapstone, while the hanging wall is generally a hard hornblendic rock. The emery occurs as lenses about 100 ft. long and averaging about 5 ft. wide. These lenses pinch out both laterally and vertically only to reappear again. The emery occurs as crystals forming about 50 per cent. of the filling of these pockets. The hanging wall stands well, so that no timbering is required except to roof over the levels and small sprags to support stagings on which they work out the pocket.

The vein has been opened up by a main adit at the level of the mill, and this cuts the vein about 200 ft. from its portal. It then follows the

vein for a distance of several hundred feet. The adit raises were carried up in the pockets of ore and the pockets were worked out by overhand stoping. Several winzes have been sunk below this adit level, and it is said that the bottoms of several of these are in good ore. There are several diamond-drill holes throughout the mine, but diamond drilling failed to furnish much information owing to the patchy character of the deposits.

Both double-jacking and machine drilling were used. Owing to the extreme hardness of the ore, a hole 3½ or 4 ft. deep was a day's shift for two men double-jacking, while the machine drills generally accomplished a little more. In a certain drift, the double-jackers were said to have driven faster than men on a machine drill, due mainly to the rapidity with which the machine dulled starters. Probably if a larger machine and higher pressure of air had been used the drill would have made a much better showing. This extreme hardness of the ore makes the mining very expensive.

At present the Ashland company is only picking over the old dumps. This cessation of mining is not due to the fact that the mine is worked out, for there is ore still left in the old pockets which can be gouged out, besides a good pocket of ore averaging 50 per cent. emery left in the Wright mine, which can be mined and delivered to the mill for less than $7 per ton. The manager of the Chester property of the Ashland company, Joseph Andrew, expects to begin mining again in the summer of 1906, and if the company were fortunate enough to find a few larger pockets, instead of small ones as has been its luck the last few years, the mines would again flourish.

There are two drifts on the vein above the adit level. The ore was loaded into cars on these levels and run to surface, where the ore from the upper drifts is dumped into chutes, from which it is again loaded into cars and run to the mill. But most of the ore was mined through the main adit level. The upper mill of the Ashland company is only 300 ft. from the mouth of the adit.

The Ashland company has two mills. The one near the mine is used at present simply for crushing and roughly washing the ore, while the other mill, half a mile from the upper one, is used for the final washing and sizing of the emery in preparing it for market.

The general plan of treatment at Chester is as follows: The ore is crushed by jaw-crushers, then it is sent to a series of rolls for further reduction; it is then washed in circular pans in which revolve two wheels called "chasers." These are built up of maple or beech blocks held together by iron straps. They are rotated by the revolving of a vertical shaft driven by belt from an overhead shaft. Ahead of each chaser is a plow, one of which moves the ore from the center to edge, while the

other moves it back again. The washer does not do any grinding, but only mulls the ore around so as to remove any foreign material adhering to the emery grains. The ore is shoveled into these washers and a little water added. After a short mulling, more water is added and the washing goes on for two hours. These washers are 6 ft. in diameter and the pulp lies about 10 in deep in the pan. The chasers are 2½ ft. in diameter. A pan will treat about 600 lb. of emery at a charge. The wash-water, carrying the foreign materials in suspension, overflows through a spout on the side of the pan and is taken in launders to a circular pan, in which revolve a series of arms which stir up the pulp and allow only the larger particles of the emery in the overflow to settle. The wash-water then goes to a rectangular settling-box, through which it is made to pass by means of partitions back and forth until it has passed over about 90 ft. of settling surface. In this first settling-tank all the flour emery is caught. The washed ore is shoveled from the pans into heaps on the cement floor; after most of the water has drained off, the emery is shoveled into coils of steam pipes in a box, the bottom of which is a screen which allows only the dry emery to run through it.

The flour emery from the settling-tank is dried on a steam-heated plate. The overflow from the first settling-trough goes to a second trough which acts as a guard to the first. This catches very little emery and is only cleaned out occasionally.

The dried emery is then passed over a series of shaking screens on which it is divided into different sizes. These screens have a throw of about 1½ in. and are actuated by a revolving cam having four arms. Wire screens are used for the sizes above 36 mesh, but those below that mesh are sized upon bolting-cloths.

Each size, after it has been screened, is taken to an exhaust blower, where it falls through an upward current of air which removes the light particles. The emery then falls upon a screen, where it is re-sized, and any oversize, due to any tear in a screen, is removed. At the Hamilton mill the emery passes twice through an upward current of air, but at the Ashland, only once. The emery from the blower is ready for market.

The ore treated at the Ashland mill contains a little magnetic material. Before going to the blower, therefore, the sized material passes through a simple magnetic separator. The ore falls in a stream past the end of a horizontal magnet core, and the magnetic particles are deflected sufficiently to fall upon an apron leading to a barrel, while the emery grains fall vertically into another. The magnetic portion is small in amount, and the pole plate is scraped clean from time to time with a knife. This magnetic material is sold for beveling glass, for it is still abrasive owing to fine particles of emery in it.

The upper mill of the Ashland company is run by steam, but the lower

one is run by a Bradley horizontal turbine with an auxiliary steam plant to use when the water power gets low. The Hamilton mill also is run by a Bradley horizontal turbine. These mills run only day shifts of ten hours. Both are well equipped and kept in good condition.

The Ashland company produces about 35 tons of finished Chester emery per month. This does not keep the mill busy all the time, so it is treating some alundum for the Norton Manufacturing Company, of Springfield, Mass.

The Hamilton company treats about 1000 tons of foreign corundum ore in a year, about half Greek and half Turkish, and containing 90 per cent. corundum.

FLUORSPAR.

BY F. JULIUS FOHS.

ARIZONA, Illinois, Kentucky and Tennessee continued to be the fluorspar-producing States in 1905. The production for the year was 50,340 short tons. Of this southern Illinois is credited with 29,059, western Kentucky, 20,281, central Tennessee 339 and Arizona 400 tons. Of the 1905 production, about 15,000 tons were ground No. 1 fluorspar, about double the amount ground in 1904. Shipments from Illinois possibly equaled two-thirds of the production, while ground fluorspar shipments from Kentucky exceeded 5000 tons as against 3421 in 1904, while the total of all kinds shipped from Kentucky was 18,872 (within 1409 tons of the amount produced) compared with 12,978[1] for 1904. Tennessee shipped 520 tons in 1905, compared with a total shipment of 600 tons for all previous years. It will thus be seen that shipments kept step with increased production.

The following table shows the number of companies that produced and prepared fluorspar for the market and the number of companies shipping to consumers in 1905. One of the shipping companies was not itself a producer.

District.	Producing.	Shipping.
Castle Dome District, Ariz.	1	1
Southern Illinois District	4	4
Western Kentucky District	12	4
Central Tennessee District	1	1
	18	10
Less companies shipping from two States	—	2
Totals	18	8

The list does not include ten or more which purchase fluorspar from the above companies and ship the mineral in their own name. Besides the producing companies, there were several others, developing mines, which did not sell any mineral during the year. The statistics of production are summarized in the following table:

PRODUCTION OF FLUORSPAR IN THE UNITED STATES.
(In short tons)

Year.	Tons.	Value.	Per Ton.	Year.	Tons.	Value.	Per Ton.
1896	6,000	$48,000	$8.00	1901	19,586	$113,803	$5.81
1897	4,379	36,264	7.65	1902	48,018	271,832	5.19
1898	12,145	86,985	7.16	1903 (a)	42,523	213,617	4.28
1899	24,030	152,655	6.35	1904 (a)	36,452	234,755	6.44
1900	21,656	113,430	5.24	1905	50,340	295,496	5.87

(a) Statistics of the U. S. Geological Survey.

[1] "Production of Lead, Zinc and Fluorspar in Western Kentucky." F. Julius Fohs, Chap. V, Kentucky State Mine Inspector's Report, 1903-4.

Fluorspar Mining in the United States.

The presence of fluorspar in the United States was reported as early as 1817, but it was not until 1870 that any shipments were made. Shipments were made that year from the Royal mines in western Kentucky. Southern Illinois followed two years later, and since 1880 production from its mines has been continuous. The annual output up to 1898 had not exceeded 10,000 tons. Western Kentucky mines have exceeded in shipments

FLUORSPAR OUTPUT OF THE PRINCIPAL PRODUCING COUNTRIES.
(In metric tons.)

Year.	France.	Germany.	Spain.	United Kingdom.	United States.
1896	1,940	21,603	3	400	5,445
1897	2,722	23,232	2	302	3,973
1898	3,077	23,787	5	57	11,021
1899	5,140	24,306	310	796	21,806
1900	3,430	30,310	4	1,471	19,646
1901	3,970	28,741	*Nil.*	4,232	17,768
1902	2,650	(*a*)14,177	93	6,388	47,190
1903	2,447	(*a*)13,028	4,000	12,102	38,577
1904	2,047	(*a*)13,540	(*b*)	18,451	33,069
1905	(*c*)	(*c*)	(*c*)	38,606	44,946

(*a*) Exports; German statistics no longer report production of fluorspar. (*b*) Not reported. (*c*) Not yet published.

those of any other American district, since 1901, and also those of any foreign country save Germany, until 1905, when southern Illinois surpassed western Kentucky. The Castle Dome district of Arizona and the central Tennessee district have been shipping since 1902. Their sales have thus far been small, but central Tennessee reports an increase for 1905. A small amount has been produced in development work in the last two years in central Kentucky, but as yet no shipments are reported. Fluorspar is secured as a by-product of lead and zinc in Albemarle county, Virginia.[2]

Practically all the mines at work in 1904 were operating in 1905, with a number of new ones. A number of new deposits were encountered in the exploitation of old mines, and in this respect the year showed a marked advance.

Central Kentucky District.—The Monitor (formerly Chinn) Mineral Company continues development work in Mercer county, at the Twin Chimney mine, some work also being done at the Fantail mine. At the former the shaft is being deepened; at the latter, a cross-cut is being driven, the vein consisting principally of fluorite. At the Twin Chimney, the vein averages about 4 ft. It is banded, for the most part symmetrically, fluorite, barite and calcite occurring in separate bands, varying from an inch to a foot in width. While barite forms only a very small part of the vein of the mine, there are other veins in the district that consist largely of barite, some of which will be operated the coming year. The shaft of the Twin Chimney has been sunk to 225 ft. (80 ft. below the adit), and the vein in the bottom

[2] "Lead and Zinc Deposits of Virginia." Thomas L. Watson, Virginia Geological Survey, 1905, p. 42.

is 5 ft. wide, largely fluorite with but little barite, with walls of Camp Nelson (Chazy) limestone. The veins of this section have been briefly noted in a recent report.[1]

Central Tennessee District.[2]—Two properties were operated in the latter part of the year. The Foley mine is situated about eight miles west of Carthage in Smith county. Here, near the surface, massive crystalline fluorspar occurs in boulders weighing from 10 to 1500 ℔. in a limy clay. In sinking the shaft, 25 ft. of limestone was encountered, immediately below which is an 8-ft. vein of fluorspar dipping 10 to 20 deg. west and striking northwest. Crystallized calcite, very similar in appearance to the fluorspar, lines the foot-wall, which consists of a gray lime-sand or clay resulting from the disintegration of the limestone. The fluorspar is said to average 98 per cent. calcium fluoride. This may be a blanket vein originally replacing limestone, similar to those in the western Kentucky and southern Illinois districts, occurring in close proximity to a vertical vein.

The Alcorn property, about 89 miles east of Nashville on the Tennessee Central, was also developed. Here yellow crystallized fluorspar occurs covering the ground within a foot or two of the surface. About 30 tons were mined from surface picking. Sinking has failed to locate the veins though fluorspar is found to fill limestone crevices 8 to 10 ft. Similar fluorite is reported from the vicinity of Nashville.

Southern Illinois District.—The Farview mine was sufficiently developed by the early part of 1905 to make it capable of turning out 50 tons daily, from its 211- and 270-ft. levels, giving it the best on record for daily production. This mine is capable of producing large amounts of fluorspar, since it has maintained a width of 6 to 30 ft. for over 500 ft. along the fault without showing any evidence of decrease. The product shows no change in the lower level. A sufficient amount of galena is secured with the fluorspar to pay more than the running expenses of the mine and mill.

The Rosiclare output was about 25 tons daily while running, secured from the 300-ft. level, where the vein shows no change in character beyond the usual pinches and swells.

At the Black mine near Bay City, where a number of small veins had been previously opened, a 12-ft. vein carrying fluorspar and galena was encountered, placing it on the list of producers for 1905. Some shipments were made from the Lee mine. Here there is an exceptionally wide vein of gravel fluorspar.

Western Kentucky District.—The widest vein operated in this district during 1905 was 36 ft. of gravel fluorspar, at the 145-ft. level of the Pogue mine. The widest lump vein was 16 ft. at the John mine. At the Mary

[1] "The Lead and Zinc-Bearing Rocks of Central Kentucky." Arthur M. Miller, *Bulletin* 2, Kentucky Geological Survey, 1905.

[2] "The data relative to the Foley mine and Alcorn property were furnished by C. D. Roberts, of the operating company.

Belle mine, two new veins were encountered and one of the shafts showed 15 ft. of fluorspar with galena associated, while another shaft at the same mine showed nearly 3 ft. of solid galena. At the John mine a 12-ft. vein of fluorspar was secured at a fault on the 247-ft. level, while within 50 ft. an 8-ft. sheeted zone running well in lead and zinc sulphides was secured.

Among the new producers was the Kentucky mine, which has a 12- to 20-ft. vein of gravel fluorspar in the upper levels and 10 ft. of lump in the lower one; this mine together with the Matthews, with which it is connected, produced a good tonnage during the year. The new Memphis slope gave as much as 12 ft. of No. 1 lump at a depth of 75 ft. The Keystone mine with a good vein carrying some galena, the Wheatcroft and a new shaft on the Tabb land may be mentioned among the others as having large veins producing in 1905, while veins of 6 ft. or less were numerous. Among new deposits may be mentioned those opened on the Ben Belt, Brown, Cox, Senator and Parish properties. The last named is a large flat deposit replacing limestone near a fault.

The shafts for the most part continue shallow, the majority of deposits being worked within 100 ft. of the surface. But a few shafts exceed 200 ft. and only the John mine has reached 250 ft.

Detail surveys were made by the Kentucky Geological Survey during the year, which I had in charge, and a deal of new and valuable data relative to the deposits and industry have been brought to light and will be published at an early date by the Survey. Two preliminary railroad surveys were executed with a view of putting a belt road into the mining field.

Castle Dome District.—This district, while on the list of producers, has not made any shipments for the last two years. It is reported[1] that there are thousands of tons of crude fluorspar on the dumps. It is secured here as a by-product from silver-lead ore. Only such fluorspar is removed from the mine as is necessary to facilitate the mining. That previously shipped from this district, it appears, was largely used in the manufacture of portland cement by California plants. Its use for this purpose has been discontinued; the lack of demand for the Arizona product may thus be accounted for, there being at present no other industries in the immediate section for which it may be used.

Fluorspar Milling.

No fluorspar milling is being done, except in the western Kentucky and Illinois districts. Some lump was shipped in previous years to be ground elsewhere, but now the ground product is largely shipped direct to the consumer. The year witnessed the largest tonnage ever known of ground fluorspar produced and shipped.

In southern Illinois, the Fairview mill was remodeled in the early part of

[1]Private communication from Robt. D. Luce, Supt., Castle Dome Mining and Milling Company.

the year, and the capacity increased. Cooley jigs and a Wilfley table were installed to replace jigs of the Harz pattern. The lead concentrate of the plant is considerably cleaner than that made previously, but the fluorspar product is not improved, a No. 2 product being turned out, crushed in two sizes. At the time of my visit, the daily capacity of the mill for concentration of fluorspar and galena was about 40 tons, while the grinding division handled about 100 bbl. per day. The capacity, however, is being increased.

No changes are to be noted for the Kentucky or the Rosiclare mill. The Nancy Hanks mill erected by the Albany Mining and Investment Company has just been completed. It is at its Nancy Hanks mine near Salem, Ky. This mill, like most of the others of the district, follows somewhat the Joplin pattern, but differs in the installation of a rather complete sizing system using a new type of shaker screen and a separate four-cell jig for each size. The problem is to concentrate the fluorspar and galena. A new grinding mill is being erected by the Keystone Lead and Zinc Company at the John mine, about eight miles from Salem, Ky.

The new Eagle mill near Salem was practically completed in 1905, and the building for the Sanders mill is about ready for the installation of machinery. The aim of these mills is the separation of the blende from the fluorite so as to produce a commercial zinc ore, and at the same time save the associated galena. The Eagle is a dry concentration plant using Hooper pneumatic tables and a very complete sizing system.

The Central Kentucky district[1] is to have a new mill, to be used for the separation and grinding of fluorspar, barite and calcite. The Monitor Mineral Company has let a contract for the mill, the work to begin on it in February. Its equipment is to consist of a picking table, two crushers of the Blake type, three sets of Cornish rolls, two four-cell differential jigs, sizing screens, accessory elevators and conveyors, a Ford concentrating table, one rotary dryer, one Griffin mill (with provision for two) to grind the fluorspar, and eight buhr mills of Virginia buhrstone for grinding the calcite and barite. An additional section of the plant is to be constructed to manufacture barium salts. Later, the calcite in part is to be utilized in the manufacture of hydrated lime and sand-lime brick at the mine.

Uses.—Among the more recent applications of fluorspar to new uses may be mentioned: (*a*) In the reduction of aluminum from bauxite, a purpose for which the demand is likely to increase, at the same time causing a decrease in the importation of cryolite. (*b*) As a flux in smelting. (*c*) As a bonding for constituents of emery wheels. (*d*) For carbon electrodes, increasing their lighting efficiency and at the same time decreasing amount of required current.

The various uses of fluorspar may be summed up as dependent on its

[1]These data were given by Prof. C. J. Norwood, Director, Kentucky General Survey.

chemical composition, fluxing properties, phosphorescence upon heating, optical and pseudo-gem properties. During 1905, between 150 and 200 companies in America used fluorspar. The increased consumption of the mineral has been noteworthy. The outlook for a largely increased demand for fluorspar in 1906 is especially bright. The iron and steel industries will require considerably more, for advance orders booked for these products at the beginning of the year were especially large. There is evidence also of increased demand for other purposes.

FULLER'S EARTH.

THE production of fuller's earth in the United States has been increasing rapidly in recent years. The major part of the output is derived from Florida. Arkansas and Alabama, Massachusetts, Colorado and NewYork produce comparatively small quantities.

Fuller's earth is a material which contains 50 to 63 per cent. silica, about 10 per cent. alumina, from 5 to 10 per cent. water, together with iron, lime, magnesia, and other impurities. In appearance it resembles clay, but when placed in water it falls into a powder, and does not become plastic.

In the United States, the principal use for the substance is for clarifying oils, especially lard and cotton-seed oil. For this purpose, a small quantity of fuller's earth, say 2 to 3 per cent., is ground to pass a 100 or 120-mesh sieve, and is added to the hot oil. After thorough agitation, the hot liquid is filter pressed, the filtrate being a clear oil, nearly white. The consumption of fuller's earth for this purpose is rapidly increasing. The statistics are given in the following table:

STATISTICS OF FULLER'S EARTH IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Year.	Production.		Imports.	
	Sh. Tons.	Value.	Sh. Tons.	Value.		Sh. Tons.	Value.	Sh. Tons.	Value.
1896......	11,326	$68,476	(a)	......	1901......	14,112	$96,835	12,061	$80,697
1897......	17,195	92,398	(a)	......	1902......	14,100	109,980	15,135	102,580
1898......	15,553	87,365	9,355	$81,044	1903......	20,693	190,277	17,100	120,671
1899......	13,620	81,900	11,558	69,460	1904......	29,480	168,500	10,221	74,000
1900......	11,813	70,565	9,154	64,790	1905......	21,745	157,776	15,181	105,997

(a) Not reported.

The most important producer of fuller's earth in Florida is the Owl Commercial Company of New York. The Southern Fuller's Earth Company, a new concern, erected a plant in 1905 for mining, drying and grinding the earth from a property near Mt. Pleasant, Gadsden county, Fla.

GARNET.

By D. H. Newland.

The production of this abrasive in the United States in 1905 amounted to 3694 short tons, valued at $114,625, an increase of 742 tons over the output of the preceding year. New York, Pennsylvania and North Carolina contributed, in the order named.

PRODUCTION OF GARNET IN THE UNITED STATES.
(In tons of 2000 lbs.)

Year.	Short Tons	Value.	Year.	Short Tons	Value.
1896........	2,440	$85,400	1901........	4,444	$158,100
1897........	2,261	66,353	1902........	3,722	122,826
1898........	2,882	82,930	1903........	4,413	146,955
1899........	2,565	72,672	1904........	2,952	89,636
1900........	3,285	92,801	1905........	3,694	114,625

The production of garnet for abrasive purposes is a small industry that has its basis in a limited and somewhat special demand of the domestic market. Outside of the United States, the industry seems to be undeveloped, though this can hardly be due to lack of deposits, for garnet is a common rock constituent. In hardness, garnet is inferior to corundum and carborundum. Its relative cheapness and adaptability to special uses, however, have furnished a field for its employment in which it has now become too firmly established to be displaced without a material change in trade conditions.

The output of garnet in New York State during 1905 amounted approximately to 2700 short tons, or 300 tons less than in the previous year. The chief producers, as heretofore, were the North River Garnet Company and H. H. Barton & Sons. The former company has opened a new deposit on Thirteenth lake, Warren county, six miles southwest of North River, where it has erected a large mill for crushing and concentrating the rock. The old workings in the town of Minerva, Essex county, have been abandoned. The deposit is of different type from the one formerly operated, the garnet being found in a hornblende-feldspar rock that probably belongs to the anorthosite series. The rock at the Minerva mine, as well as in the neighboring deposits of Gore mountain, which are operated by H. H. Barton & Sons, is a dark, hornblende schist or amphibolite and has the appearance of an altered sediment. The North River Garnet Company is the only producer that separates the garnet mechanically, the others sorting it by hand.

Some experimental work was done by G. W. Smith, of Keeseville, during the year on a deposit situated in northern Essex county. The garnet is

massive and occurs in irregular bodies in anorthosite. Except for admixture with small crystals of pyroxene the deposits are quite pure, so that there would be little waste in working. A face of almost solid garnet 30 to 40 ft. high is exposed in one place. This garnet has a loose granular texture, readily breaking up under slight pressure into small grains.

Methods of Extraction and Preparation.—Garnet is mined entirely by open-cut work and was formerly picked out by hand. By this method only the very richest beds could be worked, the decomposed surface portions usually determining the extent to which the deposit could be developed. The best garnet in the solid rock was left and covered over by the débris resulting from the working of the surface material. Since 1899, however, the North River Garnet Company has operated on a new plan, by which the rock is broken down by steam drills, crushed, and the garnet concentrated by gravity. By this method garnet 95 per cent. pure is obtained, an increase in purity of from 25 to 45 per cent. over the hand-picking method. The fact that the difference in specific gravity between the minerals to be separated is less than 0.5 makes this degree of concentration remarkable. Specimens of pure garnet and pure hornblende from the North River Garnet Company's mines gave specific gravities of 3.2 and 3.7 respectively.

Uses.—In the leather and wood industries the ready cleavage of garnet causes it to be regarded as a valuable abrasive; owing to its brittleness along cleavage planes it continually presents sharp cutting edges. The various grades of garnet paper are known commercially as sandpaper, garnet paper or shoe paper.

GEMS AND PRECIOUS STONES.

The only gem minerals produced with any regularity in the United States are beryl, garnet, peridot, ruby, sapphire, spodumene, tourmaline and turquoise; and of the States, California, Montana and North Carolina show the most activity. The following details of gem mining in the United States are taken mainly from the works of George F. Kunz, of New York, published by the California State Mining Bureau, and in the "Mineral Resources of the United States," and from those of Joseph Hyde Pratt, published by the North Carolina and the United States Geological Surveys.

Beryl.

California.[1]—The Mack mine, near Rincon, San Diego county, has yielded a quantity of fine gem beryl, and also a deep blue variety of the mineral. The beryl occurs in a 5 ft. pegmatite vein between granite and diorite. The best crystals are found in soft, clayey pockets. Another promising locality lies four miles northeast of Ramona. Here the beryls are found as solitary crystals in pockets scattered through the decomposed feldspathic central zone of a pegmatite vein. The stones are a brilliant rose pink and quite notably flawless. One perfect gem, weighing 30 carats, has been cut. Near the Mexican border, around Jacumba Hot Springs, numerous openings have shown beryl associated with garnet, in payable quantities; the region, however, is not easily accessible and is devoid of both fuel and water.

New Hampshire.—An old locality at Springfield, Sullivan county, has recently afforded some beryls of gem quality. A parallel vein, only 25 ft. distant, was formerly worked for mica.

North Carolina.[2]—Extensive mining for beryl has taken place in the vicinity of Hiddenite, Alexander county, since 1881. The beryls occur with finely crystallized quartz and rutile in the pockets of quartz veins, which cut gneisses. The productive veins all have a uniform strike nearly east and west, and dip to the north; other irregular veins show no pockets. The largest beryl ever found here was a doubly terminated crystal, of light-green color, weighing 8¾ oz. Small beryls, of deep green color, have been mined in a pegmatite vein on Crabtree mountain, Mitchell county. Fine beryls, of the aquamarine variety, are commonly found in the pegmatite veins of North Carolina, that have been worked for

[1] "Mineral Resources," 1904, p. 977 *et seq.*

[2] Joseph H. Pratt, "The Mining Industry of North Carolina during 1904."

mica. The Wiseman mine, near Spruce Pine, Mitchell county, operated by the American Gem and Pearl Company, of New York, is the largest producer. Its stones include aquamarine, the yellow and the blue varieties of beryl, in pieces such as would cut gems of about 10 carats.

Utah.—An unusual variety of beryl has been found in the Dugway range, 35 miles southwest from Simpson Springs. The crystals have a rich raspberry red color, due probably to the appreciable amount of manganese that they contain. They are tabular prisms, 3 mm. high and 7 mm. across.

Garnet.

The mining of garnet for use as an abrasive has been described on a previous page. Of the several gem varieties of garnet, the only ones in regular supply in the United States are essonite (pale yellow), pyrope (deep transparent red), rhodolite (brilliant rose), and spessartite (purple). The common abrasive garnet is mainly of the alamandite variety.

In southern California essonite and spessartite are found in significant amounts in pegmatite veins. The Surprise and the Hercules mines, 4½ miles northeast of Ramona, are the most developed prospects, while the region around Jacumba Hot Springs is no less well supplied with garnets. The Hercules mine has yielded spessartite in flawless stones weighing from 1 to 6 or 8 carats, and selling at $20 per carat. The natural conditions at this locality are less forbidding than at many others in southern California.

North Carolina has supplied gem pyrope from the gold placer workings of Burke, McDowell and Alexander counties. In Macon county the streams heading on Masons mountain have yielded beautiful rhodolites, some of which have been cut into gems of as high as 13 carats each. The rose tint of this stone is most striking in artificial and by reflected light. This is a comparatively new species, originally identified from this locality and contains nearly equal proportions of iron and magnesium oxides, besides the usual alumina and silica. The productive workings are owned by the American Gem Mining Syndicate of St. Louis.

Peridot.

The most recently discovered, as well as the most productive source of this mineral (common enough in non-gem quality in basic magnesian igneous rocks) is in Gila county, Arizona, near Talklai. The crystals occur in their native matrix, a lava, from which the weathering of its vesicular portions, frees the more resistant crystals. A single stone 1½ in. long and weighing 1½ oz. has been recorded. Numerous gems, weighing from 1 to 5 carats, have been cut from these crystals.

Ruby.

Montana.—Corundum gems of the true ruby color have been found in

association with sapphires in southern Granite county, near the headwaters of Rock (or Stony) creek. Sapphires, colorless, and also of pale-green, yellow pink, and bluish colors, constitute much the larger part of the gem wealth in this and other parts of Montana, and will be referred to under the next heading.

North Carolina.—Rubies of fair color have been found in the Corundum Hill mine, referred to on a previous page. The most favored locality for rubies, however, lies around Cowee creek, Macon county, six miles north of Franklin. Here the American Prospecting and Mining Company of New York has been working, on its 5000 acres, since 1895. The rubies are contained in gravel beds, overlain by 2 ft. of soil, and these are worked by ordinary hydraulic methods with sluices, sieves and rockers. No basic magnesian rocks, such as enclose the main corundum deposits of North Carolina, are found in the vicinity of Cowee creek, nor any limestone, although in every other respect the ruby deposits resemble closely those of Burma. The country rock is a fine grained gneiss, mainly highly decomposed; no rubies have been found in the unaltered rock. Many of the Cowee rubies contain minute inclusions which give a cloudy appearance to the polished gem. Some cut stones have weighed 3 or 4 carats and many of the smaller ones are perfectly transparent, though injured by cracks. In color and brilliancy the Cowee rubies rival those of Burma, but the output shows a rather smaller proportion of flawless stones. They show a marked pleochroism, being deep red, of the true pigeon-blood tone, when viewed along the crystallographic axis, but pale pink when seen from the side. The two most common crystal habits of these rubies are a tabular rhombohedron and a hexagonal prism.

Other localities at which rubies of gem value have been found in North Carolina are the Mincey mine on Ellijay creek, Macon county, and at a point near Montvale, Jackson county.

Sapphire.

Montana.—This is the only State to show organized sapphire mining; the output comes from three distinct localities: (1) Along a 12-mile stretch of the upper Missouri river, 15 miles northeast of Helena; (2) on Rock creek in southern Granite county, 30 miles west of Anaconda; (3) at Yogo gulch, in Fergus county, 75 miles northeast of Helena.

1. Sapphires are found in the gravel banks along the headwaters of the Missouri; the most active mining has been at the two bars named Spokane and Eldorado. The gravel beds vary from 10 to 50 ft. thick and rest upon slate, rising to 50 ft. above the river. They have been worked in turn by English and American companies, but with no financial success, owing partly to over-capitalization but particularly to the less esteemed colors of the stones. These are generally of pale-green or

greenish-yellow colors, while the prized red or blue stones are rare. For this reason, it is doubtful if the sapphires in the remaining gravel will ever pay the expense of recovering them. The crystals in this locality hold uniformly to a prismatic habit.

2. Sapphires are found in a limited area among the tributaries of Rock creek. Of the total output of 400,000 carats of rough stones recovered in the active period of 1899–1900, only 25,000 carats were fit for cutting. These stones show a wider range of colors than those from the first locality, greenish-blue being the prevailing color, not, however, approaching the prized deep-blue color of the Oriental stones. Many beautiful yellow sapphires, one of them having been cut to a 2-carat gem, are found here, as are also pink ones.

3. The Yogo gulch locality is the most widely known. Here the alluvial deposits have been exhausted, and mining is now directed to the two parallel dikes that formed the original matrix of the sapphires. These are 800 ft. apart, in limestone, and are a dark basic rock consisting mainly of biotite and pyroxene, and related to the minettes and shonkinite of the region, having, however, no feldspar. Mining is accomplished by shafts and open cuts. The upper portions of the dikes are thoroughly decomposed, rendering a simple hydraulic treatment possible. The more solid rock in depth has to remain exposed to the decomposition of the atmosphere for a season, before it can be thus treated. It is economically impossible to extract the small yield of sapphires from the fresh rock. The prevailing color of the stones is a bright blue, but a few show the dark blue of the Ceylon sapphire. In richness and brilliancy, however, they excel the Oriental stones, and are equally brilliant whether in natural or artificial, transmitted or reflected light. They differ from the prismatic crystals of the first two mentioned localities, in that they adhere uniformly to the rhombohedral habit. The largest stones found here weighed 11 to 12 carats in the rough, and 5 to 6 carats when cut. One stone of 4 carats was valued at over $75 per carat. The two operating companies at this locality are the New Sapphire Mines Syndicate and the American Gem Company. Their entire output is shipped to London. The first mentioned company has an annual output of 100,000 carats, and the second employs regularly a force of 15 men, at Helena, for cutting the gems.

North Carolina.—Small sapphires of nearly all representative colors have been found in the Corundum Hill mine, previously referred to. The part of the mine in which the gem stones are most abundant has not been worked for 14 years, but stones can be obtained by washing the alluvial gravel below the mine. This is the only locality in the United States that has yielded the emerald-green sapphire, the true Oriental emerald, and here it is rare, the yellowish and light-green varieties being

more common. What is probably the finest true emerald in the world, a crystal 4x2x1¼ in., part of it transparent, came from this mine.

Spodumene.

The usual appearance of this lithium-aluminum silicate is a massive, opaque dull white crystal, worthless as a gem; its opacity appears to have resulted from chemical alteration, a core of transparent mineral often being found at the center of a large crystal. It is found at several points in New England and at the Etta tin mine of South Dakota, where it is mined and shipped for the manufacture of lithia. Near Pala, San Diego county, Cal., spodumene is found in unaltered, transparent, and flawless crystals of a pink or lilac color; this variety has been named kunzite. The spodumenes are found in pegmatite veins, in which the quartz crystals reach an immense size; the muscovite is often replaced by lithia mica, and colored tourmalines appear in place of the usual black ones. The feldspar is decomposed, in layers, and in these the kunzites are found, singly or in pairs, embedded in the clay filling of small cavities. The presence of manganese minerals in the vein suggests manganese as the coloring agent of the kunzite.

The natural crystals range in weight up to 31 oz. troy as the maximum, and are etched and corroded on their surfaces. Pleochroism, colorless to lavender, is marked. They are readily cut into exceedingly brilliant gems of from 1 to 150 carats, selling for about $6 per carat. Their characteristic tints, pink and lavender, are among the rarest colors known in gem stones. From the scientific aspect, kunzite is particularly interesting for its behavior under the influence of ultra-violet light waves, X-rays, or radium emanations. Subjected to any one of these, kunzite shows marked phosphorescence, which, in the case of X-rays, endures for several minutes after the excitation has ceased, and with sufficient strength to produce an autophotograph on a sensitive plate.

Another gem variety of spodumene is hiddenite, in bright green transparent crystals, found at Stony Point, Alexander county, N. C. The mine has not been worked for several years.

Tourmaline.

California.—Gem tourmalines have in late years been found abundantly in southern California, always in association with the numerous lithia-bearing pegmatite veins. The output comes from three general districts: The Mesa Grande and the Pala regions of San Diego county, and the Coahuila region of Riverside county. The companies now actively productive are: Fano Kunzite-Tourmaline Mining Company, Hemet, Riverside county; Pala Chief mine, Pala, San Diego county; Tourmaline Queen mine, Pala; Himalaya Mining Company, Mesa Grande, San Diego county; San Diego Tourmaline Mining Company, San Diego.

The last mentioned is engaged in systematic mining near Mesa Grande, and operates its own lapidary in San Diego. Numerous other prospects are in progress of development, and although this district is handicapped by lack of fuel and water, the variety and quality of the tourmalines will give its development a strong impetus. The stones range from pink and bright red to deep blue and a peculiar greenish-blue; a stone of the latter sort, when cut, will show sapphire blue in one set of facets, and emerald green in another set.

Connecticut.—The old tourmaline mine at Haddam Neck, on the east bank of the Connecticut river, has lately shown a renewed activity. Here a vertical dike of nearly pure albite, over 50 ft. wide and of unknown depth, has been quarried for many years to supply white feldspar for the pottery and the polisher industries. In a zone 3 ft. wide on the eastern flank of the dike, where the albite is somewhat mixed with quartz, muscovite, lithia mica, garnets, etc., beautifully colored tourmalines are found. These are chiefly green, but others are pink or red, and some show different colors in the same crystal. Many fine stones have been cut, but the majority of the specimens have been collected in museums.

Maine.—Gem tourmalines are mined at Rumford Falls, Oxford county, in a 5 ft. pegmatite vein containing other lithia minerals. Their colors include green, red and dark blue, besides the colorless achroite. The stones are fine and clear and one of them has yielded a gem of 16 carats.

Turquoise.

The commercial supply of turquoise comes from New Mexico, where it is mined in the Cerrillos hills, in the Burro mountains, at Old Hachita, and in the Jarilla mountains. The turquoise at the first locality has been worked from antiquity. The mineral has been found in southern Colorado, and, in small quantity, in New Jersey, at the copper mines of Somerville.

Diamond.

Diamonds are brought to light from time to time in various parts of this country, in the glacial drift. Two companies are now engaged in mining for diamonds in California. One of them will work at White Rock hill, two miles northeast of Placerville, El Dorado county, where some stones have been found in times past. The other will work at Cherokee, Butte county. In the last 20 years over 60 diamonds have been found in the Cherokee mine, the most valuable one having been rated at $1200, but most of them ran in value from $20 to $100. Most of the stones are of a light-yellow tinge, although a few pure white ones have been found.

The similarity of some of the dike rocks of New York State to the diamondiferous rocks of South Africa has stimulated investigation as to their

possibilities. Daniel S. Martin has recently examined some of the most prominent dike rocks in New York[1]; he found that while the material composing them is practically identical with that in the South African peridotites, yet in New York the diamond is lacking. The only gems hitherto known to accompany the New York peridotites are pyrope, olivine, and topaz. It has been reported that diamonds have been found in the drift south of Syracuse, N.Y. The owner of a sand bed near that city claims to have found a good-sized diamond which he sold for $250. Topaz occurs in the drift, and it is believed the gem above referred to was an exceptionally brilliant topaz. It may be said, however, that the formation at Syracuse is likely to be diamondiferous, and it is possible that systematic prospecting operations would result in the discovery of diamonds.

An area in Elliott county, eastern Kentucky, on Little Sandy river, 30 miles east of Owingsville, has been examined with some care; the prevailing rock is said to be identical with that on the South African fields, and a few good stones have been found.

Diamond Mining in Foreign Countries.

For a number of years the diamond industry has practically been confined to, or at least dominated by, South Africa. The progress of diamond mining there is, therefore, a measure of the world's progress.

South Africa.

Orange River Colony.—The report of the DeBeers Consolidated Mines, Ltd., for the year ending June 30, 1905, shows a profit, somewhat smaller than that of the previous year, attributable both to a heavier outlay for mining, and to a slightly diminished yield of diamonds from the dirt of the largest mine. Details of the year's work at the company's five mines may be tabulated thus:

Mine.	Output of blue ground for year.	Yield per load.	Value per carat.	Value per load.	Cost per Load.		
					Mining. (*b*)	Washing.	Total.
	Loads. (*a*)	*Carat.*					
DeBeers	2,447,850	0.46	$12.68	$5.81	$1.17	$0.62	$1.79
Kimberley					1.29	0.71	2.00
Wesselton	2,068,278	0.284	8.86	2.52	0.61	0.31	0.92
Bultfontein	605,730	0.41	8.38	3.43	0.98	0.44	1.42
Dutoitspan	311,499	0.26	16.78	4.37	1.91	1.04	2.95

(*a*) The "load" occupies 16 cu. ft. and weighs about 1600 lb. (*b*) Including the cost of handling waste rock.

The total output of the mines during the year was 5,433,357 loads, of which 5,128,015 were washed; the stock of blue ground on the washing floors was thereby increased from 3,944,397 loads at the beginning to 4,249,739 loads at the end of the year.

The DeBeers mine has two shafts, respectively 2076 and 1670 ft. deep. Rock proved above the 1670-ft. level amounts to 2,560,900 loads; between

[1] "Peridotite Dikes of New York," Onondaga Academy of Science, New York, Oct. 24, 1905.

this and the 2076-ft. level the reserves are estimated at 3,376,300 loads.

Kimberley mine has 892,300 loads above the 2200-ft. level; reserves are estimated at 1,196,900 loads between this and the 2520-ft. level.

Wesselton mine is 500 ft. deep, and has 12,915,800 loads above this level. Inclined haulage has been abandoned; all the dirt is now dropped into the 500-ft. level and hoisted through the shaft.

Bultfontein mine is 600 ft. deep, and has a reserve of 13,866,000 loads above this level. Hoisting from the open mine has been stopped.

Dutoitspan mine is 750 ft. deep; its reserve above this level is estimated at 24,518,500 loads.

The year's work, after allowing liberally for redemption and depreciation, yielded a net income of £1,949,099, out of which £1,800,000 was distributed to holders of both common and preferred stock, at the rate of 40 per cent. each.

The Transvaal.—The report of the Premier Diamond Mining Company for the year ending Oct. 31, 1905, shows, as compared with the previous year, an enlarged output of stones, increased working and general expenses, and a diminished yield of diamonds per load of material mined. The company mined 2,036,782, and washed 1,388,071 loads, an increase of about 50 per cent. over 1904. The output was 845,652 carats, valued at £994,687, as compared with 749,635 carats, worth £866,030, in 1904. The average yield per load fell from 0.798 carat in 1904 to 0.609 carat in 1905, due to the unavoidable inclusion of a low-grade reef in the working face of No. 1 mine; much of the material washed during 1905, furthermore, came from a cut made to connect Nos. 1 and 2 workings, most of which was in barren material. Mining costs per load rose from $0.632 to $0.674 in the latter year, influenced by deeper working and a larger amount of development, charged to mining. The net profit for the year was £622,634, or £45,104 less than the profit earned in 1904, the decrease being accounted for by disproportionately augmented charges for office management and directors' fees. Dividends amounting to £260,000 were distributed from the earnings of 1905. The enormous Cullinan diamond which weighs 3 024¾ carats, discovered in the Premier mine in January, 1905, has not yet been disposed of, but is kept in stock at a nominal value of £3 290, which low valuation, in itself, creates a strong reserve.

The big diamond is a portion of a much larger stone, the original form of which can only be roughly guessed at. Four pieces of this original stone have been broken off along cleavage planes, which have the position of octahedral planes. Each of these fragments must have been of considerable size. Consequently the stone itself shows only a portion of its original natural surface (called "nyf" in the diamond-cutters' jargon), the greater portion being formed by these four flat cleavage planes. The remaining part of the surface shows one octahedral face and a curved irregular surface

roughly corresponding to six faces of the dodecahedron, while one very irregular face of the hexahedron is indicated by quadrilateral impressions which are characteristic of these faces in minerals, such as the diamond, which possesses the octahedral mode of growth.

The stone is a single crystal, no twinning planes or twinning lamellæ being present. It is quite colorless, its perfect transparency being best compared to that of pure ice or of the variety of silica known as hyalite. There are a few grains (inclusions), and also some flaws, or internal cleavage planes—"glessen," as the diamond-cutters call them—in it, but their position is such that they do not detract from the value of the stone as a gem. It is certainly the purest of all the very big stones known.

This great diamond has now reached London; previous to shipment it was insured at $1,250,000. It weighs 3024¾ carats or 1.7 ℔. troy; the largest stone previously found in the same mine weighed 392 carats only. Its dimensions are roughly 4 in. x 2½ in. x 1¼ in. Until this find was made, the biggest South African diamond was the Jagersfontein stone, which weighed 971¾ carats.

South America.

Brazil.—The export of diamonds from Brazil is undergoing a steady decline, having had a value of $315,360 in 1902, $247,042 in 1903, and $139,205 in 1904.

The diamond fields are in Bahia, and their geology has recently been described by Orville Derby in the September, 1905, *Boletim* published by the Secretary of Agriculture, for the State of Bahia. An American company is preparing to work the alluvial deposits for gold and diamonds, with dredges and other improved means.

British Guiana.—Exports of diamonds from British Guiana in 1905 amounted to 5288 carats, valued at $30,658, as compared with 11,046 carats, worth $85,947, in 1904. The stones are small, averaging, in 1905, only 0.06 carat per stone, and of a value of $5.80 per carat.

The Diamond Market.

The prices of diamonds have followed, within recent years, a steeply ascending path, in the fixing of which the two South African companies are instrumental. Three advances of 5 per cent. each occurred in 1905 and four in 1904, reaching a total of 60 per cent. increase in the London price of uncut diamonds within four years. The purchasers of rough diamonds were furthermore required to accept arbitrarily constituted lots, containing certain proportions of inferior stones, or of large stones, which do not cut to the best advantage.

The circumstances alleged by the London agencies in justification of the advances are shortages in those varieties of stones most in demand, higher

mining expenses, and the double taxation of the industry, which is levied upon both in South Africa and in England. A more probable reason is that the two formerly rival producers now see the advantage of harmony, and are together profiting by the present heavy demand for diamonds, a feature that has always been characteristic of an era of business prosperity.

The Elmore Process for Diamond Recovery.

The extraction of diamonds from the decomposed rock in which they occur in South Africa has been for a long time effected by greased tables, to which the diamonds adhere, while all other minerals are washed off. When the grease, which is of a particular kind, is removed and melted, the diamonds sink to the bottom of the pot. The process is of high efficiency, tests having showed that only 0.25 to 1.68 per cent. of the diamonds, by weight, passed off in the tailing from the table. The separation is effected by means of the property of adhesion. The Elmore process of oil concentration also operates by virtue of that property of certain minerals with respect to oil.

The experiments have resulted very successfully, practically 100 per cent. of the diamonds being recovered, and the process has now been adopted by the Premier diamond mine, which intends to erect immediately a plant capable of treating 800 tons of blue ground per day. The company is at present treating 1500 tons per day by the pulsators and grease boards.

It is thought that the Elmore process will materially reduce the cost of treatment, afford a higher extraction of the gems, and lessen the losses by theft.

GOLD AND SILVER.

By Frederick Hobart.

The gold production of the world in 1905 showed a substantial increase. The total for the year was $379,635,413, an increase of $30,547,120, or 8.8 per cent. over the output of 1904. The largest increase of the year was made in the Transvaal and, indeed, that country and the United States furnished almost the whole gain. Canada again showed a decrease. There was an improvement in British Columbia, and the comparatively small production of Ontario and Nova Scotia was practically unchanged. In the Yukon, however, there was a large decrease, due to the same causes as were noted a year ago. Mexico showed the same steady growth that has been apparent for several years. The Kolar district of India remains one of the steadiest producers. Rhodesia had a considerable increase, the result of the opening of several new mines. Australia alone among the important countries showed a decrease; this is fully explained in the more detailed report of this field, on a following page.

There was a decrease in the gold production in Russia, but less than was expected. In the Ural there was a cessation of working to a large extent, but in Siberia, from which by far the greater part of the Russian gold is obtained, the war made little difference.

The total production of gold in 1905 was not only the largest ever recorded; but it also reached an amount nearly four times as great as in 1885, only 20 years ago. There is every probability also that the increase in production will continue.

The statistics of the commercial movement of gold during 1905 show that about one-third of the year's production became immediately available in an economic sense. That is, the increase in the known bank reserves of the world was about one-third of the gold production; making allowance for certain reports which cannot be obtained, and for bullion retained in private hands, it is probable that between 45 and 50 per cent. of the gold won last year is now in use for trade purposes. It may also be remarked that the consumption of gold in the arts undoubtedly reached a very high point, as is always the case in periods of great prosperity.

Production of Gold and Silver in the United States.

The great gold output of 1904 in the United States was surpassed in 1905, and a new record was established. In 1905 nine States and territories

produced 99.5 per cent. of the total gold, the output of the remaining 12 gold-producing States being comparatively insignificant in amount. Two of the nine States produced less gold than in 1904. The other seven made gains, nearly all of them important in amount and proportion. Colorado remains the largest producer, followed by California, Alaska and South Dakota.

PRODUCTION OF GOLD IN THE UNITED STATES. (*a*)

States.	1902.		1903.		1904.		1905.	
	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value.
Alabama	119	$2,500	213	$4,400	1,417	$29,300	2,195	$46,500
Alaska	403,730	8,345,800	416,738	8,614,700	450,091	9,304,200	708,700	14,650,100
Arizona	198,933	4,112,300	210,799	4,357,600	161,761	3,343,900	169,313	3,500,000
California	812,319	16,792,100	779,057	16,104,500	924,427	19,109,600	933,142	19,168,045
Colorado	1,377,175	28,468,700	1,070,376	22,540,100	1,180,147	24,395,800	*b* 1,237,443	25,577,947
Georgia	4,730	97,800	3,000	62,000	4,688	96,900	2,441	50,500
Idaho	71,352	1,475,000	75,969	1,570,400	72,742	1,503,700	*b* 60,515	1,250,845
Maryland	121	2,500	24	500	116	2,400	281	17,000
Montana	211,571	4,373,600	213,425	4,411,900	246,606	5,097,800	245,000	5,064,600
Nevada	140,059	2,895,300	163,892	3,388,000	208,390	4,307,800	227,363	4,700,000
New Mexico	25,693	531,100	11,833	244,600	18,475	381,900	20,000	413,400
N. Carolina	4,390	90,700	3,411	70,500	5,994	123,900	3,694	76,400
Oregon	87,881	1,816,700	62,411	1,290,200	63,336	1,309,900	63,853	1,320,200
S. Carolina	5,896	121,900	4,872	100,700	5,892	121,800	4,915	101,600
S. Dakota	336,952	6,965,400	300,243	6,826,700	339,815	7,024,600	336,285	6,951,600
Tennessee	....	....	38	800	208	4 300	359	7,400
Texas	....	....	....	....	110	2,300	110	2,300
Utah	173,886	3,594,500	178,863	3,697,400	203,902	4,215,000	225,000	4,651,200
Virginia	148	3,100	654	13,500	184	3,800	19	400
Washington	13,166	272,200	13,589	279,900	15,862	327,900	17,842	368,800
Wyoming	1,879	38,800	175	3,600	793	16,400	1,485	20,700
Other States	....	....	468	9,700	....	....	....	....
Total	3,870,000	$80,000,000	3,560,000	$73,591,700	3,904,986	$80,723,200	4,260,504	$87,948,237

(*a*) The statistics in this table are as reported by the Director of the Mint, those for 1905 being the preliminary figures (subject to revision), except that later official figures for 1905 have been used, when available, as noted. (*b*) Reported by the State Inspector of Mines.

Silver production in 1905 reached a total of 58,918,839 oz., an increase of 1,132,739 oz., or 2 per cent. over 1904. Montana was the leading producer, followed closely by Colorado and Utah.

As has frequently been said, silver is now largely a by-product, obtained from ores the chief value of which is in other metals. The chief silver producers last year were the copper mines of Montana and Arizona, and the silver-lead mines of Colorado, Utah and Idaho. Production, therefore, will continue on a large scale, with little regard to the demand for silver, or its price.

In 1905 the demand for silver was very good. A prosperous season in India, combined with heavy expenditures in China, on war accounts and otherwise, to make the requirements of the Far East—always a chief factor in the market—very large. Prosperity at home and in Europe made a larger demand for the metal for use in the arts than had been known for a long time. The price of silver, therefore, was well maintained through the

PRODUCTION OF SILVER IN THE UNITED STATES. (*a*)

	1902.		1903.		1904.		1905.	
	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.
Alabama	....	$53	....	....	200	$116	387	$236
Alaska	92,000	148,760	143,600	$77,544	210,800	122,264	236,578	144,313
Arizona	3,043,100	1,612,843	3,387,100	1,829,034	2,744,100	1,571,578	3,400,000	2,074,000
California	900,800	477,424	931,500	503,010	1.532,500	144,313	1,106,772	667,937
Colorado	15,676,000	8,308,280	12,990,200	7,014,708	14,331,600	8,312,328	*b*12,831,348	7,743,718
Georgia	400	212	400	216	1,500	870	205	125
Idaho	5,854,800	3,103,044	6,507,400	3,513,996	7,810,200	4,529,916	*b* 8,626,794	5,262,344
Michigan	110,800	58,724	50,000	27,000	127,800	74,124	127,800	77,958
Montana	13,243,800	7,019,214	12,642,300	6,826,842	14,608,100	8,472,698	13,500,000	8,235,000
Nevada	3,746,200	1,985,486	5,050,500	2,727,270	2,695,100	1,563,158	6,000,000	3,660,000
N. Mexico	457,200	242 316	180,700	97,578	214,600	124,468	250,000	152,500
N. Carolina	20,900	11,077	11,000	5,940	14,800	8,584	2,547	1,554
Oregon	93,300	49,449	118,000	63,720	133,200	77,256	81,560	49,752
S. Carolina	300	159	300	162	500	290	223	136
S. Dakota	340,200	180,306	221,200	119,448	187,000	108,460	138,409	84,430
Tennessee	12,300	6,519	13,000	7,020	59,200	34,336	27,733	16,917
Texas	446,200	236,486	469,600	245,376	469,600	272,368	469,600	286,456
Utah	10,831,700	5,740,801	11,196,800	6,046,272	12,484,300	7,240,894	12,000,000	7,320,000
Virginia	5,900	3,127	9,500	5,130	6,700	3,886	4	2
Washington	619,000	328,070	294,500	159,030	149,900	86,942	115,412	70,410
Wyoming	5,000	2,650	200	108	4,400	2,552	3,528	2,152
Other States	....	....	97,400	52,596	....	....	39	24
Total	55,500,000	$29,415,000	54,300,000	$29,332,000	57,786,100	$33,515,938	58,918,839	$35,850,955

(a) The statistics in this table are as reported by the Director of the Mint, those for 1905 being the preliminary figures (subject to revision); except that later official figures for 1905 have been used, when available, as noted. (*b*) Reported by the State Inspector of Mines.

year, and in the last quarter it rose to a point higher than had been known for five years. The moderate increase in production was readily absorbed, and the conditions were favorable to the producers.

TOTAL PRODUCTION OF GOLD AND SILVER IN THE UNITED STATES.

Years.	Gold.	Silver.	Yrs.	Gold.	Silver.	Years.	Gold.	Silver.
	Dollars.	*Ounces.*		*Dollars.*	*Ounces.*		*Dollars.*	*Ounces.*
1792—1834	14,000,000	*Nil*	1883	35,000,000	35,730,000	1896	53,088,000	58,835,000
1835—1844	7,500,000	193,365	1884	30,800,000	37,800,000	1897	57,363,000	53,860,000
1845—1854	343,036,769	386,730	1885	31,800,000	39,910,000	1898	64.463,000	54,438,000
1855—1864	479,300,000	20,806,518	1886	35,000,000	39,685,513	1899	71,053,000	54,764,000
1865—1874	454,950,000	154,390,609	1887	33,000,000	41,721,592	1900	79,171,000	57,647,000
1875	33,400,000	24,533,993	1888	33,175,000	45,792,682	1901	78,666,700	55,214,000
1876	39,900,000	30,010,054	1889	32,800,000	50,000,773	1902	80,000,000	55,500,000
1877	46,900,000	30,783,509	1890	32,845,000	54,516,300	1903	73,591,700	54,300,000
1878	51,200,000	34,960,000	1891	33,175,000	58,330,000	1904	80,723,200	57,786,100
1879	38,900,000	31,550,000	1892	33,000,000	64,900,000	1905	87,948,237	58,918,839
1880	36,000,000	30,320,000	1893	35,955,000	60,000,000			
1881	34,700,000	33,260,000	1894	39,500,000	49,500,000	Total	2,786,014,606	1,620,117,161
1882	32,500,000	36,200,000	1895	46,610,000	55,727,000			

Note.—To the end of 1872, the statistics are those of R. W. Raymond, United States Mining Commissioner; subsequent statistics are those reported by the Director of the Mint.

Gold and Silver Mining in the United States in 1905.

Alaska.—This Territory, now the third producer among the States and Territories, showed an increase of 57.5 per cent., due largely to the opening and operation of the Tanana and some minor districts. The output of the Tanana country in 1905 was $4,864,000. The Nome district remains a large

producer, having turned out $4,600,000 during the year. The receipts of gold bullion at the United States assay office at Seattle during 1905 included: From Nome, 222,565; Tanana, 235,324; other Alaska regions, 46,319; and Yukon Territory, 416,432 fine oz., having a total value of $19,029,629, of which, however, $8,607,629 must be credited to the Canadian Yukon. The silver contained in this bullion amounted to 23,726 oz. from Nome, 36,097 oz. from Tanana, 6885 oz. from other Alaska, and 99,126 oz. from Alaskan and Canadian Yukon.

The Douglas island mines continue to supply the whole output of lode gold.

Alaska Treadwell.—During the year ending May 15, 1905, this mine sent 876,234 tons of ore to the mill, at a cost of $0.96 per ton including development and the mining of ore left in the stopes. The average of 4716 ore samples was $2.65 per ton. Reserves of positive ore at the end of the year were 3,959,404 tons. This mine has two mills, one of 240 stamps, operated by steam or water, and one of 300 stamps, operated only by water. During the year, the former lost only five days and crushed 396,094 tons, or 4.58 tons per stamp per day. The latter lost 79 days and crushed 481,076 tons, or 5.60 tons per stamp per day. The total recovery in bullion was $970,462 ($1.1064 per ton of ore milled); and from sulphurets, $1,037,381 ($1.1826); total, $2,007,843 ($2.2890). Mining and development cost $841,785 ($0.9597); milling, $133,547 ($0.1522); sulphuret treatment, $133,253 ($0.1519); all other operating, $36,075 ($0.0411); construction and repair, $41,365 ($0.0472); total operating and construction expenses, $1,186,025 ($1.3521); showing a net operating profit for the year of $909,-439 or $1.0368 per ton milled, including certain miscellaneous receipts. This profit is nearly 40 per cent. of the assay value of the ore.

Alaska Mexican.—During the year ending Dec. 15, 1905, this mine sent to the mill 233,985 tons of ore, averaging $3.20 in value, at a cost of $1.4192 per ton, including development. Reserves of positive ore at the end of the year were 841,876 tons. The 120-stamp mill lost 11 days and stamped 233,985 tons or 5.50 tons per stamp per day. The recovery in bullion was $358,460 ($1.5320 per ton of ore milled) and from sulphurets, $342,278 ($1.4628), total recovery $2.9948 per ton. Mining and development cost $332,077 ($1.4192); milling, $66,177 ($0.2828); sulphuret treatment, $30,401 ($0.1299); all other operating, $16,788 ($0.0761); construction and repairs, $17,278 ($0.0738); total operating and construction expenses, $464,341 ($1.9845), leaving a net profit for the year of $238,620, or $1.0198 per ton of ore milled, including miscellaneous receipts.

Alaska United.—During the year ending Dec. 15, 1905, this mine sent to mill 233,480 tons of ore averaging $2.40 in value, at a cost of $1.0196 per ton, including development. Positive ore in reserve at the end of the year amounted to 1,091,065 tons. The 120-stamp mill lost 7 days and stamped

233,480 tons, or 5.43 tons per stamp per day. The recovery in bullion was $284,875 ($1.2202 per ton of ore milled) and from sulphurets, $153,260 ($0.6564); total recovery, $438,135 ($1.8766). Mining and development cost $238,069 ($1.0196); milling, $81,407 ($0.3487); sulphuret treatment, $28,036 ($0.1201); other operating expense, $11,437 ($0.0490); construction, $8,133 ($0.0348); total operating and construction expenses, $367,083 ($1.5722); showing a net profit of $75,990 or $0.3255 per ton milled, including miscellaneous receipts.

Arizona.—The gold production of this Territory is obtained chiefly in the smelting of copper, a review of which industry has been presented on an earlier page.

California. (By Charles G. Yale.)—The gold yield of California in 1905 was $19,168,045, and the silver yield, 1,106,772 fine ounces. This shows a slight increase in the gold production as compared with the previous year. Even so small an increase was, however, quite unexpected; in fact, it was thought that there would be a large decrease.

During the later fall months a large proportion of the more important quartz mills and mines were compelled to close down owing to lack of water for power; and the hydraulic mines were unable to start working for many weeks beyond the usual time when a water supply is available.

No special developments occurred in the quartz mines of California during 1905. In the older districts work continued as usual, with the exception of the period when the lack of water compelled the companies to stop their mills. In some cases the mills were idle for two months or more, and in other instances for shorter periods. At certain points the larger companies have provided auxiliary steam power plants to be used under just such circumstances, and, after the experience of this year, it is probable that other companies will follow the example.

The southeastern portion of the State is now attracting more attention from prospectors than other portions. This is largely due to the success achieved in the "desert" camps of southwestern Nevada. Many prospectors continued southward, over the boundary line into California, prospecting the mountainous and desert regions of Inyo county and thereabouts.

As a result not only have many new finds been made, but old and practically abandoned districts have been resuscitated. Twenty-five and thirty years ago there were many camps in those sections, but owing to high costs, lack of transportation facilities, distance from reduction works, etc., most of them proved failures. General conditions have changed materially since then, and ores which could not be worked profitably then are now of value. Moreover, the experience on the deserts of Nevada has shown what may be done under adverse circumstances. The Funeral range in Inyo county has this year shown prospects of a surprising nature; and mines for years worth practically nothing in such old camps as Panamint, Cerro

Gordo, and others, have suddenly attracted attention and become of value.

In the northwestern counties of the State also, Shasta, Trinity, Siskiyou, etc., renewed activity is apparent both in quartz and gravel mines, and considerable new development is under way, especially in quartz properties. In Shasta county, the smelters purchase fluxing ores, and thereby give oportunity for many, working in a small way, to dispose of their low-grade ores. In Siskiyou and Trinity more attention is being paid than hitherto to quartz mining, these being considered generally gravel-mining counties. Siskiyou is now producing more from her hydraulic properties than any other county in the State. These counties are free from the incubus of the Caminetti law, and do not come under the supervision of the California Débris Commission; they may hydraulic as they please without having to impound the débris.

In the older central and Mother Lode counties, increased developments are to be noted in the important properties. Depths in shafts are being greatly increased; the reduction facilities for ores are being materially augmented as underground developments warrant.

Dredging operations probably attracted more attention than any other form of gold mining in California in 1905, because of their comparative novelty and rapid increase in number. The main feature of interest is their capacity for working auriferous deposits hitherto unworkable with a profit by other known methods. It is an absurd error to suppose that these machines may work either hydraulic or drift properties, as these are understood in California. The hydraulic gravel banks are usually of a depth or hight and of a grade in value far beyond the limitations of any dredge yet devised. The usual "drift" mines have their available gravel buried under several hundred feet of hard lava, which can not be handled except by means of long tunnels and drifts. Dredging machines can not manage the gravel now worked by either of these systems. But there are auriferous deposits, shallow, and with no available fall or dumps, hitherto unworked, for which the dredges are admirably adapted.

If the bedrock is not rough and hard and the deposits do not exceed say 65 ft. in depth, the overlying gravel may be readily handled with a profit, provided the average gold content is sufficient. Moreover, ground which has already been once mined may be again handled by the dredge system as at Oroville, where only the underlying deposits near bedrock were once partly mined by drifting before the dredge was invented. Moreover, ground covered with débris from former hydraulic mining, as on the Yuba river near Marysville, and useless for anything else by reason of lack of fall and water supply, can be dredged.

Comparatively shallow gravels, with an excess of water, and with soft bedrock, form ideal places for dredges to work. The costs per cubic yard vary from 2.36c. and 3c. to 8.5c., and the material thus far handled on a

large scale varies from 10 to 25c. in value, a general average being about 17c. The capacities of the machines run from 3000 to 4500 cu. yd. per day according to size and power. Generally speaking, any gravel that may be picked may be dug by the dredge.

There are at present somewhat less than 50 dredges in operation in California, but more are under construction or contemplation. The area of available dredging ground at present known or prospected is somewhat less than 50,000 acres. The principal seats of dredging operations are at Oroville, Butte county, between Smartsville and Marysville in Yuba county, and near Folsom, Sacramento county; but there are dredges at work in other counties. The average gross output per dredge, taking large and small together, is a little less than $55,000.

Colorado[1], (By George E. Collins.)—The output of the State, as a whole, showed a decided advance over that of 1904. It is true that there were no sensational new discoveries, but steady progress was made in nearly all the districts, owing largely to the improvement in labor conditions.

The output of Cripple Creek was close to $18,000,000; of this perhaps one-third was contributed by lessees, and the remainder by men working on company account. The Portland mine, of course, maintained its position of supremacy; followed by Stratton's Independence, the Golden Cycle, Findley, El Paso and Vindicator. Negotiations for the joint driving of a new drainage tunnel from Window Rock, at a level approximately 1100 ft. below that of the El Paso tunnel, are under way; it is hoped that the work will be commenced early this year.

Much interest has been aroused in the milling question, both in the direction of fine-crushing and raw-cyanidation of the lower-grade ores, and in roasting-cyanidation of the ordinary milling ores, in competition with the chlorination plants. The Dorcas cyanide mill has for several years past successfully justified its claim to existence; it is now reported that the Telluride mill will be remodeled and operated on ore from the Golden Cycle and associated properties. It is also announced that the Stratton's Independence Company proposes to build a large mill for the treatment of its dumps by the Cassell method.

Opinions differ as to the tonnage of low-grade ores ($5 to $10), now available in the mines of the district; but, in any case, it is probable that the mining and milling of what have hitherto been regarded as unpayable ores is destined to offset part of the diminution of output to be naturally looked for as some of the bonanza mines approach exhaustion.

The tonnage of shipments from Leadville increased during 1905, and will in all probability show a still larger increase during the present year. The principal shippers were the Coronado, A. M. W. and A. Y. & M. groups,

[1]Reference should also be made to the articles on "Lead" and "Zinc," elsewhere in this book.

of the Western Mining Company; the Moyer, Yak Tunnel, Ibex and Reindeer. The great orebodies in the Coronado are yielding nearly 250 tons daily; this tonnage will be doubled when connection is completed with the Penrose shaft. The Reindeer during the early part of the year produced a nearly equal quantity, but has now fallen off somewhat; the Ibex maintained a large output, mainly mined by lessees.

The rise in silver was of great benefit to Leadville, where much of the ore is of so low a grade that an addition to the margin of 50c. to $1 per ton means all the difference between loss and profit; mining costs there have unquestionably reached the lowest point (about $1.40 per ton) ever attained in this State, under parallel conditions.

As a whole the Leadville district is now in excellent condition for extensive production; the ore now proved insures its prosperity for some years to come.

In the San Juan district the production of Ouray county was largely maintained by a single mine, the Camp Bird; the annual report of this shows that its average monthly output is nearly $200,000. The work of unwatering the old Red Mountain bonanza mines makes good progress; the Joker tunnel has already unwatered the upper workings of the Guston; great hopes are based on the revival of the area.

At Telluride the principal mines, the Smuggler-Union, Liberty Bell and Tomboy, appear to be entering upon a period of prosperity. The first named has been leased in large units, each of which forms virtually an extensive mine; both mills are running at full capacity. Recent developments in the Pennsylvania tunnel (Smuggler-Union) and Stilwell tunnel (Liberty Bell), at about the same horizon, justify the expectation that the principal mines in the district have a long life ahead of them. The production of the Liberty Bell was curtailed by difficulties in connection with the reconstructed mill plant; this has been remodeled as a fine-sliming and filter-pressing plant. The Alta is being operated by a leasing company.

In the Silverton district, the Silver Lake, Gold King and Sunnyside continue the principal mines. New milling plants are being erected for the Neigold, Mogul, Gold Prince, and other properties. The year was on the whole a prosperous one, although the bulk of the material treated was of a very low grade.

In Boulder county, mining is now more active than for many years past. The improvement is due partly to reasonable smelting rates but partly also to realization of the fact that success is more likely to follow the adoption of standard methods than of the theories of process mongers. Much interest is being taken in the results of experiments in cyaniding telluride ore after roasting, which in many cases, as might have been expected, are entirely successful. Boulder county is never likely to rival the principal

districts of the State, as the veins are comparatively narrow; on the other hand, one may safely predict a larger output in the future.

Conditions in Gilpin county have been somewhat depressed for two or three years, owing mainly to the exhaustion of local capital and the lack of outside assistance, which have curtailed development. Several properties, however, made excellent records during 1905, among which the Running Lode and Old Town should be mentioned. As usual, many parties of lessees have done well, especially in the East Notaway, Golden Wedge and Old Town. The depression is particularly noticeable in the Central and Nevada districts; at Russell Gulch the reopening of the Pewabic and the prosperity of the Old Town secure employment for every available miner; great things are hoped from the unwatering of the Saratoga by a lateral from the Newhouse tunnel.

In the Clear Creek district development was active; there was an ample supply of outside capital, thanks to a number of energetic promoters. Some of the mines, such as the Little Mattie, Gem and Lamartine, employed many lessees, most of whom did well. At Idaho Springs the community pins its faith to the deep-level tunnels, prominent among which are the Newhouse, Central and Lucania (projected to drain the Quartz Hill and Russell districts of Gilpin county, and to secure the transportation of the ores produced therefrom), and the McClelland tunnel, which is planned to secure the same advantages for the Freeland district.

Above Georgetown the Argentine district will be greatly benefited by the rise in silver and by the improvement in the lead and zinc markets. The new owners of the Dives-Pelican and Terrible have erected mills for the treatment of the low-grade silver ores in the dumps and on the stulls of these famous old producers; preliminary runs are stated to give satisfactory results. Important underground discoveries are said to have been made in the Terrible; on the other hand, the production from lessees in the Silver Plume district has fallen off.

The silver-producing districts of Aspen and Creede have had a fairly prosperous year. At Aspen, the Smuggler was the largest shipper; the total shipments were larger, in quantity at all events, than for several years.

Of the independent smelters, the Golden smelter, which was largely supplied from Clear Creek and Gilpin counties, closed down during the summer, and it is hardly likely to reopen under present conditions. The "pyritic" smelters at Silverton and Ouray were not operated during the year; but a similar plant has been built at Grand Junction, and is said to be at work. The only independent smelting plant (aside from the Argo) in successful operation is that at Salida; this is mainly supplied from the New Monarch mine at Leadville, but it has also important contracts with mines in the Cœur d'Alene district.

As a whole, the immediate outlook is satisfactory. Cripple Creek may be expected to maintain its present rate of production, while Leadville and the San Juan will probably show a considerable increase. The recent rise in silver, if maintained, will be of great advantage to most Colorado mining districts; for although it is not likely to lead to any considerable increase in production, it will add materially to the margin of profit. A revival of intelligent prospecting is urgently needed; in facilitating this the extension of the Moffatt railroad through Routt county may prove of great benefit, not only to the northwestern part of Colorado, but also to the State at large.

Idaho. (By R. N. Bell.)—The gold production of Idaho during 1905 showed a decrease, which was due to several causes, among them being a very light snow fall and short water season with the placer mines of the State, whose yield still amounts to nearly one-half of the total output; and also to the suspension of milling operations at two of the most important quartz gold producers—the Kittie Burton mine in Lemhi county, whose fine 30-stamp mill was burned in September, and the DeLamar, the reconstruction of whose mill in Owyhee county was begun in the summer, and is now nearly completed.

The losses from these two sources amounted to fully 10,000 oz. The new DeLamar mill, however, is about completed, and should soon begin operation again with a daily capacity of 100 tons. The product of this mine averages about $10 per ton, of which something like 80 per cent. is gold and 20 per cent. is silver. It is successfully treated by direct cyaniding, and a high extraction made at the low cost of $2.40 per ton. The ore reserves of the mine, of the grade mentioned, are extensive.

What seems likely to prove one of the most important gold developments of the year was at the property of the Crooked River Mining and Milling Company, at Orogrande, near Elk City in Idaho county. This property carries a remarkably interesting deposit that is locally called a dike, but is really a sheeted, shattered and excessively fissured zone of the ordinary country rock, which is a coarse gneiss, near a great field of eruptive granite. This zone is from 300 to 500 ft. wide, and presents an appearance of a rusty, shattered, granitic gangue that has evidently been permeated by gold-bearing iron solutions; it averages better than $3 per ton in gold. This material has been mined by the glory hole method. The milling equipment consists of a 20-stamp mill and a 300-ton cyanide plant, which is in successful operation at the present time. What the exact results are cannot be learned, but from the appearance of the material it should yield readily to cyanide treatment, as it is of a friable nature, and completely oxidized to a depth of from 100 to 200 ft. Below that horizon, the ground changes to a blue granitic gangue, quite evenly permeated with fine-grained iron pyrites, but no other objectionable

sulphides are manifest. Some smaller fissured courses are included in this great zone, carrying much higher values than the average, and milling results from these selected streaks are said to run as high as $15 to $20 per ton. The property promises to become a producer of the Treadwell class, as it has a great lineal extent, and is admirably situated for economical handling. Before the cyanide plant was installed, some extensive amalgamation tests were made with the 20-stamp mill, and mining and milling costs were reduced to 60c. per ton. There are several other deposits of the same class in the Elk City basin that are worthy of extensive development.

Elk City also contains a pronounced series of nearly vertical gold-bearing quartz fissures, 5 to 10 ft. wide, that carry bunches of high-grade tellurium ore, associated with specks of gray copper and iron pyrites at the deepest points of development, which, however, are all comparatively shallow. Several carload shipments have been made of first-class ore from these smaller fissure veins that have yielded gold returns of from $100 to $200 per ton. The American Eagle mine was operated two years ago with a light 10-stamp mill, and produced $75,000 in bullion. Development on the Buster mine has recently disclosed some very rich ore, which is also true of the South Fork mine and several others. Elk City is an old placer district that has made an important total yield since its discovery, amounting to several million dollars, and still contains some extensive tracts of flat gravel beds that may be profitably dredged.

The Sunnyside mine, at Thunder Mountain, which was equipped with a 40-stamp mill early last spring, has proved a disappointment so far, as the best ore obtainable from the mine, in any quantity, did not pay running expenses, and the property of the Thunder Mountain Gold and Silver Mining and Milling Company, locally known as the Dewey mine, is still the only producer of this much advertised district. Its output in 1905 exceeded $60,000; the mill was undergoing repairs part of the time and was operated for only eight months of the year.

A new State wagon road is under construction to the Big Creek district, 40 miles north of Thunder Mountain. This district has some wide lodes or zones of gold- and silver-bearing rock, upon which considerable development has been done, and is likely to receive more attention from investors during the coming season, when the wagon road is completed.

In Lemhi county, the plant of the Kittie Burton Gold Mining Company is being rebuilt, and the property will enter the list of important producers again during the coming season, as it has large reserves of good ore developed. There are some extensive deposits of $8 to $10 gold ore, at Silver Creek and Singiser, and two of the properties are equipped with new mills, recently completed, with a combined capacity of 100 tons per day.

One of the most important new sources of gold during the coming season will be from the operation of dredges in the Boise Basin district. A new electric power-plant and dam are being constructed on the south fork of the Payette river for the purpose of supplying electric power to the dredges and other mining enterprises of the Boise Basin; they are expected to be completed and in running order in time for the placer season of 1906. There are four large dredges in the Boise Basin; two of them are of modern design and construction, and two more will be built next season according to definite plans already under way. This famous old placer district has some extensive tracts of flat gravel beds that make ideal conditions for dredging; they have been tested and operated to some extent, in a practical way. They vary from 10 to 40 ft. deep and contain average values ranging from 10 to 40c. per cu. yd., and are sure to afford an important increase in the gold production of Idaho, commencing with next season.

The chief source of the silver output of Idaho is the Cœur d'Alene mines, yet the Trade Dollar Consolidated mine at Silver City in Owyhee county continues an important producer and made an output of 800,000 oz. silver and 5000 oz. gold during 1905. This mine is equipped with a 20-stamp mill, and already has a bullion record of over $12,000,000. It is opened on a narrow vertical fissure that cuts two heavy flows of igneous rock, rhyolite and basalt. Its lower levels are in eruptive granite. The principal feature of its development is a drain tunnel, 11,000 ft. long, with a maximum depth of 1700 ft. and extensive intermediate levels and connections. Recent developments have added ore reserves that will supply the present milling capacity for several years. The property is ably managed. Its power requirements are transmitted electrically from the company's own plant at Swan falls on the Snake river, 25 miles distant. There are several other important mining developments in progress in this old district, and promising evidences of new bodies of rich silver-gold ore are manifested.

Almost all the lead deposits of Idaho carry silver values equal to, or higher than, those of the Cœur d'Alene, and their further extensive development should account for a steadily increasing yield of silver from this State. This is also true of some extensive deposits of silver-gold, silicious milling ores that are undergoing steady development and equipment with milling facilities in the Atlanta district of Elmore county, the Yankee Fork district, of Custer county, and the Singiser, Silver Creek and Parker Mountain districts of Lemhi county, and it is not unlikely that within a couple of years the silver output of Idaho mines will exceed ten million ounces.

Montana.—Four-fifths of the silver output of Montana comes from the copper mines of Butte, reference to which is made under "Copper" on

an earlier page. In the productions of gold, however, the Butte copper mines are second in importance to the gold mines of Fergus county, where the cyanide practice prevails.

Fergus county has the chief gold producers in Montana. The output of the mines, including gold, silver and lead, was valued at about $1,200,000. The chief center of activity is Kendall, 18 miles north of Lewistown. The principal mines are on North Moccasin mountain; they include the Kendall, Barnes-King, North Moccasin and the Bullard mines. The ore occurs in the form of a blanket vein. A large tract of new ground (known as the Fergus tract), which extends from the flat country up to the foot of the mountains, is being actively prospected by diamond drills.

In Lewis & Clark county active development work is being done on the famous Whitlatch-Union mine. Regular shipments are being made. The Jay Gould mine has been rejuvenated, and affords employment to 100 miners. The Howard mine is also on good ore. Shipments now being made to East Helena average $30.

Beaverhead county, once noted for its gold production, produced very little in 1905. Bannack placers were almost deserted. The dredges on Grasshopper creek were idle. The chief producers in 1905 were the various placers on the Grasshopper and Rattlesnake creeks.

The Southern Cross mine (in Deer Lodge county, 22 miles west of Anaconda) had a successful year. It shipped nearly $1,000,000 worth of gold ore. The ore is an auriferous gossan and is used extensively at the Anaconda smelter in fluxing the Butte ores.

During the year the Granite-Bimetallic company, of Philipsburg, Granite county, pulled the pumps out of the mines and quit business. The cause of suspension was indebtedness due to attempts to discover fresh orebodies.

A district which attracted considerable attention is the Little Rockies, where several mines gave excellent results during 1905, although almost unknown two years ago. This district covers an area about 12 miles square, in the eastern part of Choteau county, the extreme northern county of the State, and comprises the larger part of the Little Rocky mountains. It is 50 miles south of the Great Northern railway and 25 miles north of the Missouri river, and can be reached by stage from Malta and also from Harlem. The principal town is Zortman on the east side of the mountains. About 125 men are now employed at the various mines and mills in the district.

The orebodies are large and strong and are found in deposits along the contact between limestone and porphyry and in veins and replacement masses within the porphyry itself. Some of the ores resemble those of Cripple Creek, Colo., and Mercur, Utah, and those of the Moccasin mountains, Fergus county, Mont. When found in the limestone near the contact

with porphyry the ore is a brecciated and decomposed limestone, containing broken pieces of hard limestone colored by iron oxide, and by a purple mixture of fluorite and quartz. This is soft and easily mined and lies in lenticular masses along the contact, or, in other instances, conforms to the formation and stratification of the limestone. This character of ore is the main supply of the district. Some of the largest orebodies are, however, entirely within the porphyry along a series of fissures in it. In this case the ore is porphyry in which the feldspar has been replaced by quartz, the iron being oxidized near the surface and the porphyry very much altered. Deeper down the ore is altered porphyry and quartz with a percentage of iron sulphide.

There is always more or less silver with the gold, but no free gold. Like the ores of the Kendall and Gilt Edge districts these are particularly adapted to treatment by cyanide processes, and there are two cyanide plants in operation at Zortman. One of these belongs to the Ruby Gulch company, and has 120 tons capacity; this during the past summer sent to the assay office at Helena about $20,000 per month. This mill is being remodeled to give greater capacity. That of the Alder Gulch company is of 100 tons capacity and during the past summer's run gave returns of about the same amount as the Ruby mill. Both these properties are being more thoroughly developed. The Papoose Mining Company will build a 300-ton mill and cyanide plant in 1906.

Nevada.[1] (By A. Selwyn-Brown.)—The Tonopah mine was discovered in May, 1900, and in 1901 became productive. The high returns obtained by the lessees (under the difficulties usually experienced in remote camps in desert regions) soon attracted attention. In the latter part of 1901 the rush to the district began. This led to the opening up of the whole of the southern portions of the State, as well as the southeastern portion of California. Prospecting operations are still being pursued over a wide area and rich discoveries are frequent.

Tonopah.—This is the best established and most promising camp in southern Nevada. The Tonopah, Montana, Midway, Extension, West End and other mines have larger reserves than are seen in any of the newer camps. Other properties are being developed which will possibly pick up some of the known orebodies. It is a singular fact that the first developed mines at Tonopah are today the richest.

The Tonopah mine is now yielding a net profit of $3,000,000 per annum. This is being earned from an average shipment of only 850 tons a week, nearly all of which is mined by hand power from development work on and above the 600-ft. level, and raised to the surface by small gasoline hoists. For this reason, and also because the big smelting plants cannot treat a greater quantity of Tonopah ores, the output of the mine is small compared with

[1] Reference should also be made to the articles on "Copper" and "Lead," elsewhere in this book.

what it can produce. In a short time the company will have replaced its gasoline hoists with powerful electric hoisting plants at the Mizpah, Silver Top, Red Plume and Desert Queen shafts, and power drills will be used throughout the mine. Early in 1906 the company's 100-stamp mill near Tonopah will reduce about 450 tons a day.

The Montana mine ranks after the Tonopah. It has been developed to a depth of about 800 ft., and has over 9000 ft. of underground drifts and winzes. These workings have disclosed five distinct mineralized ledges. The ore reserves opened up in this mine are large. The company contemplates the erection of a metallurgical plant.

The Midway adjoins both the Tonopah and the Montana. It has been developed to a depth of 600 ft. There is a large quantity of high-grade ore blocked out between the 365-ft. and 600-ft. levels. Two rich veins have been developed, and several others which are opened in the Montana are expected to be found in the Midway ground when crosscutting is resumed.

The Tonopah Extension adjoins the Tonopah on the west. The company owns two patented claims and a fraction of another. This mine was one of the early and most successful of the Schwab-McKane investments. It has been developed by a main shaft (500 ft. in depth) and by numerous drifts. At the 200-ft. level one of the veins, starting away from the Tonopah mine, was encountered. It is 10 ft. wide and is said to average $50 per ton. A crosscut from the 500-ft. level also cut a wide ledge, carrying shipping ore.

The West End mine is south of the Tonopah Extension. The shaft has reached a depth of 700 ft. At 390 ft. a well-defined quartz ledge of great width was cut. In the 415-ft. level a large orebody was recently opened.

On Mount Oddie, west of the Tonopah and Montana mines, are situated the Belmont, North Star and Mizpah Extension mines. The veins now being worked on the Montana and Tonopah mines have been found in the Mount Oddie mines.

During 1905 Tonopah was furnished with a good water supply and with electric lighting and power. Sanitary conditions were improved; the health of miners is now more carefully watched than formerly. The monthly payroll in Tonopah amounts to $250,000.

In the vicinity of Tonopah are situated the Manhattan, Liberty, Ray, Crow Springs, Lone Mountain, Gold Mountain and other centers where mines are being actively developed.

Bullfrog.—The Bullfrog district is south of Goldfield, a little north of the Funeral Range, and extends practically to the California border line. It has come into prominence on account of the extent and richness of the gold mines on Montgomery mountain.

The Montgomery-Shoshone mine is the leading property of the district. The main shaft is 150 ft. in depth, and is in sulphide ore. The Shoshone

vein has been opened on the southwest in the Polaris and Del Monte claims owned by the Montgomery-Polaris Company. The vein appears to be running into the Bullfrog-Steinway property, where the same character of talc ore has been struck on the 100-ft. level. Several other mines are developing orebodies of promise.

Goldfield.—This town is situated 25 miles directly south of Tonopah. It is a much younger mining center than Tonopah, but its name has become familiar throughout the country on account of the good returns some of the mines yielded during 1905 to miners who worked several of the vein outcrops on lease.

The Jumbo is one of the largest mines in the camp. The deepest shaft is a little over 450 ft. in depth. This mine has been involved in litigation.

The Combination, lying to the southwest of the Jumbo, is next in importance. This is paying regular dividends of $32,000 monthly. The Florence, which lies between the Jumbo and Combination, is a property of promise; the main shaft is down about 360 ft.

Goldfield possesses five stamp-mills and two samplers, the combined capacity being about 1800 tons per week. It is proposed to increase this capacity. An important event of the year was the completion of the Tonopah railway to Goldfield. It is probable that this line will be extended to Bullfrog during 1906.

Kawich.—This is a promising district eight miles east of Goldfield. A large area has been located on the Kawich range. The Gold Reed Company recently cut a body of ore in a tunnel 150 ft. below the outcrop of a vein, which yielded high returns when first opened on the surface.

The Comstock.—Most of the celebrated mines on the Comstock are still working. Their annual output continues to be insignificant. The Comstock, however, is far from being worked out. The mines contain large bodies of low-grade ore; when present practices are revolutionized, and the field is operated on genuine commercial lines, it may again become an important gold and silver producer.

At Gold Hill, a couple of miles southwest of Virginia City, several gold mines are being successfully operated, in a small way.

New Mexico. (By Charles R. Keyes.)—Though gold was mined by Europeans in New Mexico before it was anywhere else in the United States, and though some of the most extensive and richest placers on the continent occur in this region, yet the insufficient water supply was long a drawback to the exploitation of these valuable gravels. The dredge at last has solved the most difficult of the problems; in a number of localities these machines are in operation, and with large profit. There are hundreds of localities where similar operations may be carried on, with large returns on the outlay. Near some of the best placers, the solid and unweathered rock carries over $1 in gold to the cubic yard.

Lode mining has received more systematic attention than for a long time previous. The general outlook for a considerable increase of output is promising. In the western part of Socorro county important mines are being developed, but the distance (100 miles) from railroads is a serious hindrance to extensive operations at this time.

At present, silver mining offers attractions for investment that have not occurred during a dozen years previous. With improved recent methods of treatment, and with better transportation facilities, New Mexico should soon regain much of the old-time prestige she had before silver fell so low in value.

Many of the silver and gold ores are complex, and heretofore have defied profitable treatment. Since satisfactory methods have recently been put into operation, a large amount of ore (which up to this time could not be shipped) can now find a market.

South Dakota.—The report of Mine Inspector Nicholas Treweek says that the output of the mines during 1905 was 2,080,271 tons, of which the Homestake handled 1,534,000 tons. The men employed in the mining industry total 3549, of which number 2800 are in the employ of the Homestake company. The total bullion value (gold and silver) for the year is reported as follows: Homestake, $5,080,000; Golden Reward, $391,351; Horseshoe, $379,182; Alexander Maitland, $320,000; Imperial, $251,000; Lundberg, Dorr & Wilson cyanide mill, $184,400; Spearfish, $157,919; Dakota, $120,338; Gilt Edge Maid, $110,665; Wasp No. 2, $86,325; Hidden Fortune, $80,724; Clinton, $10,705; Portland, $8905; total mines, $7,181,534. The output from placers is estimated at $10,000; making a total of $7,191,534 for the year.

The report of the Homestake Mining Company for the year ending June 1, 1905, shows that 1,398,100 tons of ore were mined and treated, yielding $5,221,089 in bullion, an average of $3.734 per ton. The income account is given in a rather involved form, but a fair statement of expenses is as follows:

Mining, $2,492,257 ($1.782 per ton of ore milled); milling, $600,685 ($0.430); cyaniding, $345,349 ($0.247); general, etc., $295,396 ($0.0211); total, $3,733,687 ($2.670), leaving net earnings of $1,568,651 or $1.122 per ton, including miscellaneous receipts.

The expenses were 70 per cent. of gross earnings. Of the expenses a total of 66.8 per cent. is charged to mining; 25.3 per cent. to milling and cyaniding; 7.9 per cent. to general and miscellaneous expenses, which include repair shops.

Developments in various parts of the mine progressed steadily and satisfactorily. The ore at the 1250-level was reached at the close of the year and cross-cutting began at once. The Ellison shaft was sunk to 1400 ft., and the work of opening a level at that depth begun. The B. & M.

shaft has attained a depth of 1250 ft.; the Golden Prospect, 900 ft. The Old Brig and the Golden Gate remain at 800 ft. each, and the Golden Star at 1100 ft.

(By J. V. N. Dorr.)—During 1905 the Black Hills saw considerable progress along conservative lines, with eight companies operating their own reduction plants continuously, and several others shipping to custom mills and the smelters outside the Hills.

The overproduction of mining stocks, from which the Hills suffered several years ago, has left some of the more speculative propositions apparently stranded, but the producing properties have, as a rule, operated steadily with diminishing working costs.

The formation of the Consolidated Power and Light Company will reduce the power cost at the mines and mills in the Northern Hills materially and will assist greatly in the development of properties not on the railroad.

This company has taken over the lighting plants of Deadwood and Lead, the latter of which was already supplying power to two mills, and is now installing a modern steam-turbine power plant of several thousand horsepower capacity. The investment of such a large sum of money as this company will expend shows confidence in the permanency of the mining industry in the Hills.

The Homestake company increased its tonage milled considerably over 1904; the last annual report showed an increased recovery per ton milled. Early in the year the dividend rate was doubled, and it is believed that the present rate will be maintained, although the company has improvements under way that will require the expenditure of more than $500,000 in less than a year. These improvements comprise the erection of a filter-press plant to treat the 1300 tons of slime now being discharged without being cyanided; the installation of a power plant to generate 500 h.p. from the water supply for the mills; and the substitution of compressed air for animal haulage underground.

The Homestake is the only steady producer in the free-milling district at present. The Clover Leaf company (which has been operating for the last few years against great difficulties from large amounts of water) has shut down to secure additional capital; and the numerous companies floated on the "extensions" of the Homestake have not yet passed the development stage.

Most of the companies operating in the "silicious ore" district did well. Both the Wasp No. 2 and the Golden Reward company increased the capacity of their mills, and the Puritan company is putting up a wet-crushing mill to handle quartzite, similar to that crushed dry in the Wasp.

The burning of the big mill of the Horseshoe company, in May, shut down the largest producer; but the mines have been operated steadily since

then, shipping high-grade ore to the smelters and a lower grade of ore to a custom mill. The reorganization of the company, now about completed, will result probably in the changing of the old dry-crushing plant at Pluma into a 300-ton wet-crushing mill.

The purchase of the McGovern ground in the Portland district by the Imperial company, at the end of 1904, has proved a good investment, and that company supplied its mill with good ore from that property during the whole year.

There was considerable activity in the Limestone district. The Spearfish ran steadily all the year, and put in another set of rolls in order to grind finer; the Victoria and Eleventh Hour companies are each completing a 200-ton mill similar to that of the Spearfish company.

The only new company to start milling during the year in the northern Hills was the Gilt Edge Maid company.

In the southern Hills a number of properties are being developed. The Golden West company ran its mill at Rochford until the shortage of water for power purposes in the fall shut it down.

Utah.—A large part of the gold production of Utah is recovered in the smelting of copper ores, the mining of which has already been reviewed under "Copper," and of lead ores, which will be mentioned under "Lead." Among the producers of gold alone, Mercur remains preëminent. Development at the gold camps in southwest Utah is being stimulated by the opening of the new San Pedro, Los Angeles & Salt Lake Railroad to through traffic.

During the year ending June 30, 1905, the Consolidated Mercur mined and milled 245,026 tons of ore, averaging $3.95 in value. Of this tonnage, one-third was base ore, and two-thirds was oxidized. Much of the mining during the year was directed towards the opening of old caved stopes, preparing to mine them by the caving system. In the scheme of ore treatment, the leaching of slime has been abandoned. The average extraction during the year was $2.97 per ton, an improvement of 11c. per ton over that of the previous year; loss in tailing was exceedingly variable, but averaged $0.98 per ton, a slight reduction from the preceding year. Mining and prospecting cost $1.51 per ton of ore, an increase of $0.11; cyanide treatment cost $1.12 per ton, a reduction of $0.48; total, except construction, $2.63, which is the lowest operating cost that the mine has ever accomplished.

Gold Mining in Foreign Countries.

Africa.

Egypt.—Prospecting continues successfully in the Nile Valley and the Eridia. In the Nile Valley mine, the main shaft has been sunk to

200 ft. The reef is 5 ft. wide and carries free gold; a trial crushing of 150 lb. of selected ore returned 218 oz. of gold.

The Wm. Garaiart mine has been vigorously developed. Five shafts have been sunk and drifts have been run on three levels. Of 10,650 tons of ore extracted and treated, 94 tons produced 9242 oz., 371 tons produced

GOLD PRODUCTION OF THE WORLD.

Countries.	1903.			1904.			1905.		
	Oz. Fine.	Kilo-grams.	Value.	Oz. Fine.	Kilo-grams.	Value.	Oz. Fine.	Kilo-grams.	Value.
AMERICA, NORTH:									
United States	3,560,000	110,730.9	$73,591,700	3,904,986	121,445.1	$80,723,200	4,260,504	132,519.6	$87,948,237
Canada	911,639	28,355.8	18,843,590	793,420	24,675.4	16,400,000	700,863	21,800.0	14,486,833
Newfoundland	6,844	212.9	141,477	6,242	194.0	129,022	4,550	141.5	94,049
Mexico (a)	515,464	16,033.1	10,654,641	620,209	19,291.1	12,819,720	702,799	21,860.0	14,526,855
Central America	90,711	2,821.4	1,875,001	54,218	1,686.3	1,120,700	(e)54,214	1,686.3	1,120,700
AMERICA, SOUTH:									
Argentina	1,451	45.1	30,000	446	13.9	9,200	(e)446	13.9	9,200
Bolivia	944	30.0	19,520	147	4.5	3,000	(e)147	4.5	3,000
Brazil	110,014	3,421.9	2,724,000	98,354	3,058.8	2,032,984	(a)117,396	3,651.5	2,426,575
Chile	45 ,801	1,424.6	946,707	30,812	958.4	636,900	(e)30,812	958.4	636,900
Colombia	131,785	4,098.9	2,724,0 0	95,520	2,970.8	1,974,400	(e)95,513	2,970.8	1,974,400
Ecuador	13,304	413.8	275,000	6,430	200.0	132,900	(e)6,430	200.0	132,900
Guiana (British)	77,939	2,424.1	1,611,000	70,661	2,198.0	1,460,580	82,300	2,559.9	1,701,141
Guiana (Dutch)	21,942	682.5	453 548	25,778	801.8	531,831	34,442	1,071.3	711,916
Guiana (French)	101,645	3,161.5	2,101,000	86,532	2,691.5	1,788,800	(e)86,532	2,691.5	1,788,800
Peru	26,910	837.0	556,230	17,406	541.4	359,782	(e)17,406	541.4	359,782
Uruguay	2,796	83.9	7,800	1,227	40.0	25,368	(e)1,227	40.0	25,368
Venezuela	14,514	451.4	300,000	(e)14,514	451.4	300,000	(e)14,512	451.4	300,000
EUROPE:									
Austria	264	8.2	5,457	2,283	71.0	47,190	(e)2,283	71.0	47,190
Hungary	108,526	3,375.6	2,243,232	117,949	3,668.7	2,438,006	(e)117,949	3,668.7	2,438,006
Germany (c)	82,690	2,572.0	1,709.202	88,029	2,738.0	1,819,518	126,446	3,933.0	2,613,639
Italy	2,029	63.1	41,939	325	10.1	6,718	(e)325	10.1	6,718
Norway	358	11.1	7,404	350	10.9	7,234	(e)350	10.9	7,234
Portugal	42	1.3	864	40	1.3	827	(e)40	1.3	827
Russia	1,208,530	37,590.5	24,980,320	1,212,055	37,700.0	25,053,177	1,063,883	33,402.3	22,197 155
Spain	260	8.1	5,382	257	8.0	5,316	(e)257	8.0	5,316
Sweden	1,640	51.0	33,900	1,958	60.9	42,235	(e)1,958	60.9	42,235
Turkey	997	31.0	20,607	1,400	43.5	29,000	(e)1,400	43.5	29,000
United Kingdom	4,545	141.3	93,953	17,537	545.5	359,719	(e)13,584	422.5	280,781
AFRICA:				65,076					
Madagascar	55,150	1,715.4	1,139,963	233,199	2,024.1	1,345,121	66,258	2,060.9	1,369,553
Rhodesia	201,960	6,280.0	4,174,513	3,779,621	7,238 3	4,820,223	348,518	10,840.4	7,203,865
Transvaal	2,963,681	92,164.3	61,259,286	94,815	117,551.4	78,124,766	4,897,221	152,324.1	101,225,558
West Coast	70,763	2,201 0	1,462,671		2,949.1	1,959,826	165,844	5,158.4	3,427,995
ASIA:									
Borneo (British)	41,000	1,275.2	847,470	42,745	1,329.5	883,539	(e)42,745	1,329.5	883,539
China (e)	314,465	9,780 9	6,500,000	217,688	6,771.0	4,500,000	217,688	6,771.0	4,500,000
East Indies (Dutch)	32,343	1 006 0	668,529	50,797	1,580.0	1,049,967	(e)50,797	1,580.0	1,049,967
India (British)	542,191	16,864.4	11,207,099	557,007	17,325.6	11,513,340	576,889	17,943.7	11,924,308
Japan (g)	100,951	3,140.0	2,086 657	217,707	6,771.6	4,500,000	(e)217,707	6,771.6	4,500,000
Korea	169,32×	5,266.6	3,500,000	(e) 48,379	1,505.0	1,000,000	(e)58,055	1,805.7	1,200,000
Malay Peninsula	15,724	489.1	325,000	18,990	590.7	392,522	(e)18,990	590.7	392,522
AUSTRALASIA: (d)	4,315,759	134,238.2	89,206,739	4,220,690	131,281.2	87,241,662	4,159,220	129,369.2	85,970,779
Other Countries	72,570	2,257.2	1,500,000	72,570	2,257.2	1,500,000	72,570	2,257.2	72,570
Totals	15,939,469	495,760.3	$329,475,401	16,888,367	525,255.0	$349,088,293	18,431,070	573 596.6	$379,635,413

(a) Figures based on exports and coinage. (c) Includes output from domestic ores only. (d) Six states and New Zealand. (e) Estimated. (f) Includes Servia, Persia, West Indies, Formosa, British New Guinea and Philippine Islands. (g) Exclusive of Formosa.

Note.—The value of gold is $20.67 per troy ounce, which is equivalent to $664.55 per kilogram.

761 oz., and 870 tons produced 447 oz. Nothing less than 0.5-oz. ore will pay. It is obvious, therefore, that the future success of the mine depends entirely on the presence of shoots and patches of high-grade ore.

The Om Nabardi mine in the Sudan developed so well that a milling plant was erected to treat the ore. Developments have been actively con-

ducted, and the presence of ore over 3000 ft. in length and varying from 1 to 4 ft. in width, assaying 0.5 to 1.5 oz., has been definitely proved. A light railway 30 miles long was built, to connect the property with the Sudan railway.

The Eridia Exploring Company was formed in London in 1903, with the consent of the Egyptian Government, to take over prospecting rights over 1200 square miles. The concession lies to the east of the town of Keneht and is 400 miles south of Cairo. It is easily reached by fairly good roads

SILVER PRODUCTION OF THE WORLD.

Country	1904.			1905.		
	Oz. Fine.	Kilograms.	Value.	Oz. Fine.	Kilograms.	Value.
AMERICA, NORTH:						
United States	57,786,100	1,797,390.6	$33,515,938	58,918,839	1,832,623.3	$35,850,955
Canada	3,718,668	115,666.2	2,127.859	5,974,875	185,843.7	3,605,957
Mexico (*a*)	67,008,448	2,084,244.1	38,342,234	70,838,066	2,203,361.2	42,750,773
Central America	664,267	20,661.4	380,100	(*e*)664,267	20,661.4	400,885
AMERICA, SOUTH:						
Argentina	67,108	2,087.3	38,400	(*e*)70,000	2,177.0	42,245
Bolivia	6,166,093	191,791.4	3.528.300	(*e*)6,600,000	205,287.7	3,983,100
Chile	879,922	27,369.2	503,500	(*e*)850,000	26,438.2	512,975
Colombia	958,914	29,826.2	548,700	(*e*)1,000,000	31,103.5	603,500
Ecuador	(*e*)40,000	1,244.2	22,888	(*e*)40,000	1,244.2	24,140
Peru	4,666,993	145,163.0	2,670,500	(*e*)5,000,000	155,521.0	3,017,500
Uruguay	1,049	32.6	600	(*e*)1,093	34.0	660
EUROPE:						
Austria	1,279,633	39,801.4	732,219	(*e*)1,279,633	39,801.4	772,284
Hungary	525,723	16,352.4	300,824	(*e*)525,723	16,352.4	317,284
France	617,955	19,221.0	353,600	(*e*)617,955	19,221.0	372,948
Germany (*c*)	12 535,238	389,898.5	7,172,817	12,535,238	389,898.5	7,565,267
Greece	855,912	26,622.5	519,200	(*e*)855,912	26,622.5	516,500
Italy	801,917	24,943.0	458,857	(*e*)801,917	24,943.0	438,973
Norway	(*e*)257,200	8,000.0	147,170	(*e*)257,200	8,000.0	155,225
Russia	172,912	5,378.2	100,300	172,912	5,378.2	104,356
Spain	3 774,989	117,418.0	2,160,049	(*e*)3,774,989	117,418.0	2,278,281
Sweden	20,923	650.8	12,113	(*e*)20,923	650.8	12,627
Turkey	572,342	17,802.2	327,500	(*e*)572,342	17,802.2	345,420
United Kingdom	159,689	4,967.0	91,374	159,689	4,967.0	96,376
ASIA:						
Dutch East Indies	177,907	5,533.6	101,800	(*e*)177,907	5,533.6	107,370
Japan	3,252,303	101,160.0	1,861,000	(*e*)3,215,000	100,000.0	1,940,153
AUSTRALASIA	14,757,170	459,010.0	8,444,200	14,362,639	446,738.4	8,667,853
AFRICA	493,001	15,334.4	282,100	493,001	15,334.4	297,536
Other Countries (*d*)	50,000	1,555.2	28,610	(*e*)50,000	1,552.2	30,176
Total	182,262,376	5,669,124.4	$104,772,752	189,830,120	5,904,508.8	$114,856,349

(*a*) Statistics compiled from export and coinage. (*c*) Silver produced from domestic ores only. (*d*) The output is mostly from China and Persia. (*e*) Estimated.

Note.—Unless specified to the contrary, the statistics have been taken from official sources. The average commercial value of silver for 1903 was 53.45c. per ounce. equivalent to $17.18 per kilogram; for 1904 it was 57.221c. per ounce, or $18.40 per kilogram; and for 1905 it was 60.352c. per ounce, or $19.40 per kilogram.

from the Nile by the Wady Hammama. At Eridia there are two distinct series of gold-bearing veins, one on the east and the other on the west side of the Wady. These veins are parallel and are 2.5 ft. wide. The ancient workings on "A" vein can be traced for over half a mile. The district is well suited for mining, as the hills are steep, and it will be easy to drive tunnels at the level of the Wady. The ore averages 0.7 oz. gold per ton.

The Fatira Exploring Company is successfully developing the Semna mine. The reef is 6 ft wide and has an average value of 2 oz. gold per ton.

Madagascar.—For several years great attention has been given by

French colonists to prospecting for gold. The exports in 1903 and 1904 amounted to 50,532 and 65,076 oz., respectively. The gold output of 1905 was 66,258 oz., valued at $1,369,553. A large number of persons are prospecting for gold, but most of them are men who have had but little experience. Some rich gravel has been found. Gold-bearing reefs and alluvial deposits have been found in various parts of the island. Early in 1905 Madagascar gold mining propositions were boomed in the Transvaal; but investigations showed that the published statements regarding the richness of the claims were inaccurate. This led to a slump in values. The largest French company operating gold mines in Madagascar is the Lyons-Madagascar Syndicate of Lyons, which was formed in 1895. Several concessions were secured, and in 1897 the syndicate became a limited-liability company. Prospecting was begun on concessions at Anasaa and Andramary and at several places in the Province of Fénerive. By the end of 1903 the company had made a net profit exceeding $20,000 out of the gold obtained during prospecting.

Rhodesia. (By W. Fischer Wilkinson.)—The gold output of Rhodesia in 1905 was 409,836 crude oz., valued at $7,203,865 (£1,480,449), which was recovered from the milling of 1,031,326 tons of ore. This is the largest output yet recorded, and the industry shows a rapid growth, as the following table indicates:

Year.	Oz. crude.	Year.	Oz. crude.
To Sept. 1, 1898........	6,471	1903................	231,872
Sept. 1-Dec. 31, '98...	18,085	1904................	267,737
1899................	65,304	1905................	409,836
1900................	91,940		
1901................	172,062		
1902................	194,169	Total to 1906.........	1,457,476

The output of silver was 70,146 oz. in 1904 and 89,728 oz. in 1905. The following analysis of the December yield indicates the source of that month's gold.

GOLD OUTPUT OF RHODESIA, DECEMBER, 1905.

	Number of:			Ore crushed.		Product from:		
	Mines.	Stamps.	Days Milling.	Total.	Per stamp per day.	Mill.	Tailing.	Other.
				Sh. Tons.	*Sh. Tons.*	*Oz.*	*Oz.*	*Oz.*
Matabeleland	48	589	222¾	63,825	4.75	20,963	4,323	389
Mashonaland...........	38	270	222¾	28,318	4.81	8,840	2,347	522
Totals.............	86	859	222¾	92,143	4.78	29,803	6,670	911
Value..................	....	....	....	....	....	110,489	22,245	3,484

The leading producers were the Ayrshire mine, which had 60 stamps and had a monthly output of £12,000, and the Globe and Phœnix, with 40 stamps and a monthly yield of £16,000. In the Gwanda district, the West

Nicholson, Geelong and Eagle Vulture mines were amalgamated into one company called the East Gwanda Mines, Ltd., which in September had 100 stamps at work, yielding 3425 oz. of a value of £11,322. The Sabiwa mine, which is also in the Gwanda district, has a 40-stamp mill and four 6-ft. Huntington mills, which will shortly commence crushing.

The Killarney mine, with 20 stamps and one tube-mill, produces from £5000 to £6000 worth of gold a month. The Selukwe mine has a 40-stamp mill and the output is worth about £8000 a month. The Wanderer is a large low-grade mine, worked as a quarry, milling ore of about 3½ dwt. grade with a dry-crushing plant equal to 100 stamps. The ore is crushed to ¼-in. mesh, and is cyanided direct. The working expenses are about 8s. per ton milled. The Surprise mine with 20 stamps, and the Penhalonga mine with 45 stamps, each make monthly returns of from £5000 to £6000. A new feature in Rhodesian mining is the large number of mines that are worked by tributers, with small mills of from 2 to 10 stamps. During September there were 829 stamps at work, of which 595 were in mills of 20 stamps and upward, and 234 in mills of under 20 stamps. There are several new mines coming on, among which are the Giant, with 15 stamps and three tube-mills, in the Hartley district; and the Jumbo (with 45 stamps projected), in the Salisbury district.

Considerable interest was occasioned by the development on the so-called "banket" formation in the Lonagunda district, which lies 75 miles northwest of Salisbury. Except for the fact that the formation contains boulders and pebbles, it bears no resemblance to the Rand conglomerate beds; to call it "banket," which, in South Africa is a "name to conjure with," is misleading. The formation appears to be a schist made up of hornblende and chlorite, containing pebbles of granite, granophyre and quartz diorite. The width of the formation varies considerably; at one place, where it outcrops in a very distinct manner on the Hanyani river, the width is about 30 ft., with the granite country rock on both sides.

The formation has been traced in an east-and-west direction for 25 miles. With the exception of the work done at the Eldorado mine, little development work has been done to prove the value of the formation throughout this length. The property has been reported on by E. H. Garthwaite, consulting engineer to the British South Africa Company; his conclusions were that while the values obtained on the Eldorado (considering the amount of work done) were very satisfactory, unduly high values have been placed on this and other properties belonging to the company.

The Transvaal. (By W. Fischer Wilkinson.)—The mining industry continues to expand, the returns again showing an increase over previous years. The production of gold during 1905 for the whole of the Transvaal was 4,897,221 fine oz., having a value of £20,802,074 or $101,225,558, the corresponding figures for 1904 being 3,779,621 oz., £16,054,809 or $78,124,766.

Although only a few new mines have been added to the producing list during the year, those that were crushing last year have in many cases increased their plant and output. The Witwatersrand mines that make returns this year and not last are the Consolidated Langlaagte, with 120 stamps; the Porges Randfontein, with 100 stamps; the Princess, with 50 stamps; the Orion, with 20 stamps; the Simmer & Jack East, with 100 stamps; and the Village Deep, with 110 stamps, the last two making their debut. The Wolhuter and Ginsberg mines lost their mills by fire during the year.

The number of stamps at work in the Witwatersrand district in December, 1904, was 5555, while there were 6930 in operation in December, 1905. The stamp basis is not, however, any longer a good standard of comparison, owing to the recent introduction of auxiliary crushing machinery, such as tube-mills, and it is desirable now to define the size of a plant by the tons it can treat. For instance, a mill of 100 stamps and two tube-mills could do about as much work as a 120-stamp mill without tube-mills, with stamps of equal weight.

TRANSVAAL GOLD PRODUCTION FOR 1905.
(Chamber of Mines Returns.)

Witwatersrand Mines.						Outside Mines.	All Transvaal.
Month. 1905.	No. of Companies.	Tons Milled.	No. of Stamps.	Fine Gold, Oz.	Value.		
January	62	835,823	5,740	357,214	£1,517,349	£51,159	£1,568,508
February	63	801,320	5,876	351,052	1,491,174	54,197	1,545,371
March	63	907,620	6,195	385,575	1,637,818	60,522	1,698,340
April	64	898,871	6,345	385,394	1,637,050	58,500	1,695,550
May	63	956,857	6,542	400,149	1,699,725	69,000	1,768,734
June	63	935,245	6,620	396,188	1,682,900	68,512	1,751,412
July	62	958,214	6,640	401,121	1,703,854	78,090	1,781,944
August	64	983,151	6,835	410,859	1,745,218	75,278	1,820,496
September	63	940,918	6,770	399,536	1,697,121	72,003	1,769,124
October	63	959,844	6,725	397,868	1,690,036	75,011	1,765,047
November	65	979,959	6,930	407,056	1,729,064	75,189	1,804,253
December	65	1,002,600	6,930	414,421	1,760,349	72,946	1,833,295
Total	....	11,160,422	....	4,706,433	£19,991,658	£810,416	£20,802,074

As in previous years, I give tables showing the monthly tonnage treated, and the gold production for the year under review, as well as for previous years. Between January and December, 1905, there was an increase of 19.95 per cent. in the tons milled, and of 16 per cent. in the gold won.

Last year I called attention to the decrease in the yield per ton; as this is a matter which is disquieting to some people, I wish again to explain that it is the natural result of the increased scale of operation; of the inclusion of poorer mines in the list of crushing companies; and of a reduction in the cost of working. In the early days it was the richest mines that came into operation first—and the average grade of ore sent to the mill was comparatively high. As larger plants were put up it was possible in many

TRANSVAAL GOLD PRODUCTION BY YEARS.
(Chamber of Mines Returns.)

Year.	Witwatersrand District.			Outside Mines Value.	Transvaal Total.
	Tons milled.	Value.	Value per ton milled		
		£	s.	£	£
1884–9	1,000,000	2,440,000	48.8	238,231	2,678,231
1890	730,350	1,735,491	47.4	134,154	1,869,645
1891	1,154,144	2,556,328	44.2	367,977	2,924,305
1892	1,979,354	4,297,610	43.4	243,461	4,541,071
1893	2,203,704	5,187,206	47.0	293,292	5,480,498
1894	2,830,885	6,963,100	49.2	704,052	7,667,152
1895	3,456,575	7,840,770	45.2	728,776	8,569,555
1896	4,011,697	7,864,341	39.2	739,480	8,603,821
1897	5,325,355	10,583,616	39.74	1,070,109	11,653,725
1898	7,331,446	15,141,376	41.3	1,099,254	16,240,630
1899(*a*)	6,639,355	14,046,686	41.14	661,220	15,728,693
1899(*b*)	233,395	1,020,787	....		
1900	459,018	1,510,131	65.82	....	1,510,131
1901	412,006	1,014,687	49.25	81,364	1,096,051
1902	3,416,813	7,179,074	42.00	74,591	7,253,665
1903	6,105,016	12,146,307	39.79	442,941	12,589,248
1904	8,058,295	15,539,219	38.46	515,590	16,054,809
1905	11,160,422	19,991,658	35.82	810,416	20,802,074

(*a*) Jan. to Oct. (*b*) Nov. to Dec., supplementary (incomplete).

cases to lower the grade, without reducing the profit. The effect of the reduction in grade has been to raise the available tonnage of a mine, and consequently to lengthen its life. The lower the grade that can be profitably worked the less waste will there be of the gold that Nature has provided. It is safe to predict that the average yield per ton milled of all the mines will gradually fall; and it should be regarded not as an unfavorable feature, but as a healthy phenomenon indicative of the expansion of the mining industry.

The cost of working the mines is gradually being reduced, mainly by reason of improvements in the methods of mining.

The conditions under which the mines are operated have not improved to any great extent since the war. It is true that explosives are cheaper; but skilled and unskilled labor (the large items amounting to from 50 to 60 per cent. of the operating costs) are much as they were. White wages remain high, owing to the high cost of living. A paper read before the British Association (which visited South Africa during the year) gave the average monthly expenses of a married man, with wife and three children, at nearly £25 per month. Supplies, excepting explosives, are, on the whole, no cheaper; in consequence of high railway rates and customs duties. The list given herewith shows the prices paid at one of the mines in 1899 and 1905 (see p. 242).

The only item which shows a substantial reduction is explosives, which have been reduced from 97s. 6d. a case in 1899, to 57s. in 1905. Explosives account for about 9 per cent. of the total mining costs, so that the saving under this head amounts to only 4 per cent. over pre-war costs. The in-

creased economy may be chiefly attributed to improved methods of mining. Perhaps the most important reductions are due to the introduction of piece-work wherever possible. This has been done with beneficial results, both to employer and employed, especially in mines working with Chinese, who prefer working on contract.

Much attention has been paid to the all-important subject of working costs. Although the majority of mines keep the most detailed accounts, yet a comparison of different periods is not an easy matter, as it necessitates

COMPARISON OF PRICES.

Articles.		July, 1899.	July, 1905.
Acid, sulphuric	per lb.	2½d	2¾d
Bolts and nuts	per cwt.	31 to 35s	25s 6d
Cyanide	per lb.	1s 0d	10.63d
Cement	per cask	45s 4d	28s to 30s
Ceiling, Oregon	per ft.	3¾d	3⅞d
" Baltic	"	3¾d	2⅛d
Candles (Crane's)	per box of 25 lb.	9s 9d	9s 4¾d to 8s 6¼d
" (stearic acid)	"	19s	16s
Detonators	per box of 100	4s (No. 6)	5s 9d (No. 8)
Deal	per cu. ft.	4s 11d	3s
Explosives, gelatine ⅞ inch	per case of 50 lb.	97s 6d	57s
Fuse	per coil	$4\frac{1}{16}$d	4d
Flooring	per ft.	5d	4½d
Iron, bar	per cwt.	16s 8d	14s 6d
Iron, galv. corrugated	per ft.	6⅜d to 7d	5¾d
" " plain	per 100 lb.	23s	22s
Karri wood	per cu. ft.	7s	7s
Lime, white	per cask	5s 3d to 6s 2d	7s 9d
" blue	"	4s 6d	3s 6d
Meat	per lb.	5d	3½d
Mercury	per flask.	£10	£8 10s
Nails	per cwt.	19s o 25s	17s 6d to 20s
Oregon	per cu. ft.	3s 9d	3s
Pitchpine	"	5s 6d	3s 6d
Paraffin	per case	13s 9d	12s 3d
Rails	per cwt.	11s 4d to 12s	12s 4½d
Screening, battery	per sq. ft.	5d	4¼d
Steel, octagon	per lb.	5⅛d	5⅛d
" cruciform	"	5⅛d	5⅛d
Zinc disks	"	5d	4d to $4\frac{1}{32}$d

an inquiry as to whether the conditions (such as the stoping widths, the labor supply, the amount written off for development, the scale of operations, etc.) are the same over the periods compared.

I give herewith figures comparing the costs for two periods of the Simmer & Jack mine; the mine was operated in one period with Kaffirs, and in the other with Chinese labor. Some allowance must be made for the fact that the number of stamps working during the two periods was not the same, but the figures would only be slightly affected by the difference. It must also be noted that during the 1905 period many of the Chinese were new arrivals and consequently inefficient. Later months show a reduction in costs, that for December being 19s. 11.707d. per ton milled.

Looking to the future it may be anticipated that the cost of working may be reduced owing: (1) To the larger scale of operations, which will tend to reduce fixed charges; (2) to the more extended adoption of contract work;

(3) to the use of small machine drills; and (4) to a reduction in the cost of supplies.

The introduction of the small machine drills to take the place of hand drilling has not yet proved a success on this field. Experiments have been made in this direction, but so far no drill of this class has been installed on a large scale on any mine.

COMPARISON OF COSTS PER TON MILLED—SIMMER & JACK PROPRIETARY MINES.

	Average Recovery	No. of Stamps.	Average Sorting
1899—April to June	33s.10.731d	280	19.8%
1905—April to June	31s. 7.748d.	320	18.5

	April-June, 1899.		April-June, 1905.	
	d.	%	d.	%
European labor	77.299	32.0	85.446	33.0
Colored labor	38.523	16.0	41.887	16.2
Compound expenses (including food and supervision)	25.722	10.7	36.993	14.3
Supplies	77.725	32.2	73.529	28.3
General	22.066	9.1	21.386	8.2
	241.335	100.0	259.241	100.0

The experiments, however, have been encouraging, and there is good reason to hope that a serviceable drill will be manufactured before long. The introduction of a good machine drill that could be used by unskilled labor in the stopes would be of great importance to the industry, as it would at once go a long way to solve the most pressing problem of the day—the dearth of unskilled labor. Supposing a good reliable machine were invented (with which an unskilled laborer could drill four or five holes per shift, instead of one as at present), the number of unskilled laborers required would at once be reduced in proportion. Especially is this of importance for the deep-level mines, where the time and expense of transporting the labor, to and fro, is a serious problem.

Chinese labor has continued to attract public interest, both in South Africa and in Europe. Outside South Africa, and even in South Africa itself, except on the Rand, the situation was little understood. Most extravagant and misleading accounts were circulated as to the behavior and treatment of the Chinese. By some, the employers were pictured as the most brutal of taskmasters; others represented the Chinese as lawless, and a danger to the community. It seems scarcely necessary (seeing that the Transvaal is a British colony) to deny that the Chinese are subjected to ill-treatment. It is generally admitted—and the mines are open to inspection to all the world—that the Chinese are well fed and well cared for, and that they live in clover here, as compared with their life in their own country. What gave rise to the rumors of ill-treatment were the so-called "desertions" of the Chinese from the mines. Owing to loose supervision, considerable numbers of the Chinese absented themselves from the mines

where they were supposed to be at work. On investigation, it was found that a large number of the absentees were loafers about the mine who did not trouble to turn up at the roll-call. Food could be obtained whether they worked or not, and hiding in some sunny corner was preferable to doing a day's work in the mine. Others again, influenced by the spirit of adventure or curiosity, wandered off into the country; it is probable that the cases of attacks on lonely farm-houses originated from hunger, and that the outrages were not premeditated. On the other hand, it is probable that the attacks on stores in the mining area were organized from motives of revenge on the storekeepers for cheating the Chinamen. At any rate the desertions were not due to ill-treatment. As soon as it was found out that many were absent from the mines, a general search was made; the loafers or deserters, the most of whom were discovered in the neighborhood, were brought back to the mines. The Chinese appear to be quite contented with the conditions under which they live. They spend their money freely, the bicycle being the most popular purchase. They have their own theaters, and present plays which apparently give every satisfaction to the audience. At one mine they have a brass band which already performs Western music very creditably. From an economic point of view the experiment has been a decided success.

The number of Chinese employed on the Witwatersrand gold mines at the end of December, 1905, was 47,217, representing 38.5 per cent. of the total unskilled labor supply in the Witwatersrand district. The Chinese are distributed over 34 mines, the largest employer being the Simmer & Jack Proprietary mine, which had in October 4309 on its books. The work done by the Chinese compares favorably with that of the Kaffirs; the mines employing Chinese are now worked as economically as the mines worked entirely by Kaffir labor. The introduction of this new class of labor was not accomplished without many difficulties. The Chinese were strange to the country, strange to the work and unable to make themselves understood. Misunderstandings were at first frequent, and accounted no doubt in many cases for disputes with white miners. Among the Chinese first imported were many bad characters; and many of the disturbances which have occurred have no doubt been due to their evil influence. Steps have been taken to repatriate the ringleaders, and, speaking generally, the work now goes on smoothly. The Chinese live in separate quarters at the mines, and are allowed to move about freely within the mining area under a system of passes.

The opponents of Chinese labor contend that the introduction of this foreign element has curtailed the employment of white men. This, however, is not the case; on the contrary it can be shown from statistics that the number of white men employed on the mines has largely increased since the introduction of Chinese.

In May, 1904, before the introduction of Chinese, the total number of natives employed on the gold mines (according to the statistics of the Government Mines Department) was 77,519, and the total number of whites 13,127. In the month of December, 1905, the unskilled labor force was: Natives, 93,831; Chinese, 47,267; or a total of 141,098; while the number of whites was increased to 18,159. These figures demonstrate that the increase in the unskilled-labor supply has largely increased the employment of white people. It must also be remembered that the increased employment of white people on the mines means also that other white people can earn a livelihood in commercial pursuits.

As regards the supply of Kaffir labor, the demand continues to be largely in excess of the supply, in spite of the most active recruiting. The total natives available for mining and "other employ," chiefly domestic service (Sept., 1905), exclusive of natives employed on farms, has been estimated at 177,061; of which number 85,634 found employment on the mines. The demand for labor for "other employ" is an increasing one.

LABOR STATISTICS (MINES DEPARTMENT). PERSONS EMPLOYED IN GOLD MINES AT END OF MONTH.

	White.	Colored.	Chinese.
1902—July	8,162	32,616	....
December	10,292	45,698	....
1903—June	11,825	66,221	....
December	12,695	73,558	....
1904—June	13,413	74,632	1,004
December	15,023	83,639	20,885
1905—June	16,989	104,902	41,340
December	18,159	93,831	47,267

The number of persons employed on all mines of the Transvaal at the end of December, 1905, was: White, 19,527; colored, 109,649; Chinese, 47,267.

TRANSVAAL LABOR, 1905.

Employed at end of each month. (Chamber of Mines.)

	Natives, Witwatersrand.	Outside Districts excluding Babbertonar.	Total Natives.	Chinese.	Total Natives and Chinese.	Natives Employed by C. S. A. R., S. A. C. P. W. D. and other Departments.
January	76,040	5,405	81,445	27,197	108,642	19,739
February	84,186	5,181	89,367	31,389	120,746	20,477
March	90,290	4,314	94,604	34,282	128,886	20,261
April	92,084	4,130	96,214	35,516	131,730	19,732
May	91,647	4,579	98,226	38,066	134,292	19,251
June	88,843	5,145	93,988	41,290	135,278	19,149
July	86,375	5,298	91,673	43,140	134,813	18,590
August	83,370	5,459	88,829	44,562	133,391	18,025
September	80,306	5,328	85,634	130,122	44,488	18,205
October	78,541	5,134	83,675	45,901	129,576	17,684
November	77,758	5,204	82,962	45,804	128,766	18,190
December	75,333	5,621	80,954	47,217	128,171	18,486

During 1905 a considerable amount of railway construction was undertaken, which will help the outside mining districts. The most important new line (which is now near completion) is that connecting the present Johannesburg-Klerksdorp line with the main trunk line between Cape Town, Kimberley and Bulawayo. The line follows the Vaal river, joining the main line, to the north, at Fourteen Streams. There is a branch line from Eastleigh Junction to the Vierfontein and Groenfontein collieries. The distance between Johannesburg and Kimberley by the new route will be shortened to 308 miles, while a saving of several hours will be made in the journey to Cape Town. Another projected line is to connect Krugersdorp, Kleinfontein and Zeerust, which will no doubt ultimately be continued to Mafeking. North of Pretoria a new line to Rustenburg is being constructed, of which about 21 miles are nearly completed. Toward the east the Johannesburg-Brakpan line is being extended to Witbank, to connect with the Delagoa Bay line. This will shorten the route for the coal traffic from the Middelburg coalfield to the Rand. A new line is also projected to connect Delagoa Bay and Johannesburg, via Bethal and Swaziland, which will provide better gradients than the Witbank-Brakpan line. This line is open for traffic as far as Klipstapel, from which place a branch line to Ermelo is being constructed.

The profits of the gold mines have allowed of dividends to the amount of £4,857,539 being declared for 1905. A table is given herewith showing the amounts distributed annually in dividends since 1887.

DIVIDEND LIST OF GOLD MINES OF THE TRANSVAAL.

Year.	£	Year	£	Year.	£	Year.	£
1887..	12,976	1892..	879,320	1897..	2,707,181	1902..	2,121,126
1888..	112,802	1893..	527,284	1898..	4,848,238	1903..	3,345,502
1889..	432,541	1894..	1,527,284	1899..	2,946,358	1904..	3,911,093
1890..	254,551	1895..	2,046,852	1900..	....	1905..	4,857,539
1891..	334,698	1896..	1,513,682	1901..	415,813		

The most important development in metallurgical practice has been in connection with finer reduction of the ore to obtain higher extractions, largely due to the success of the decantation process for the treatment of slime. Fine crushing can be carried out, either by using heavy stamps with finer screens in the battery and higher discharge, or by the use of tube-mills, or other grinders, for the coarser portion of the tailings. While the advantage of finer grinding has been demonstrated, the best method of carrying it out has not as yet been finally decided on. With a modern plant, an average extraction of 90 per cent. on ore of a value of 10 dwt. and upward can now be relied on; the development above noted is expected to raise this to 95 per cent., whereas a few years back the average extraction was between 80 and 85 per cent.

As regards the use of tube-mills, several have now been installed. They may be employed: (1) To increase the tonnage crushed, a coarser screen than usual being used in the battery; (2) to carry, where so desired, the reduction of the ore to a finer point than is economically possible in the stamp battery; (3) to slime the pyritic concentrate, when concentrators such as Wilfleys or Frue vanners are installed below the plates.

On the East Rand new mills are being constructed to work on the last-mentioned principle. Practice (1) is employed at the Knights Deep, where 200-mesh per sq. in. screen in the 100-stamp battery allows a duty per stamp per 24 hours of well over 7 tons. The coarser portion separated by spitzlutte after leaving the plates passes through two tube-mills; the final cyanide residues have an average value per ton as low as is obtained by ordinary milling practice. Practice (2) is employed at the Robinson Deep, where the pulp from the plates is hydraulically sized, the coarser sand amounting to 50 per cent. with returns passing through the two (22 ft. by 5 ft. diam.) tube-mills. The aim of the millman here is to obtain, in his final pulp going to cyanide works, as small a percentage as possible of sand coarser than 60-mesh. The work done by the tube-mills is estimated in terms of the tons reduced from plus 60 to minus 60 and the increase in minus 90 mesh. At the Robinson Deep (during August, 1905) the pulp leaving the 200-stamp battery contained 27 per cent. coarser than 60-mesh, and 54 per cent. finer than 90; the pulp overflowing the tube-mill spitzlutte contained about 6.75 per cent. coarser than 60, and 79.3 per cent. finer than 90. A table given herewith shows the comparative results on the cyanide residues of this mine, when milling with, and without, tube-mills:

1905.	Tons Crushed by Battery.	Average Value Residue, Dwt.	Screen Used in Battery.
Average Jan. to March, without tube-mills.....	26,872	1.26	1,000 light
August, with 2 tube-mills..	29,173	0.78	800
Increase in extraction......	....	0.48	

The tube-mill working costs were 1s. 0.92d. per ton ground, and 5.88d. per ton milled, giving an additional extraction of 0.48 dwt. per ton milled, or 2s., and an additional profit credited to tube-mills of 1s. 6.12d. per ton milled.

In cyanide works, a small economy is made by the use of bisulphate of soda (a by-product of the dynamite factory), in place of sulphuric acid, in refining the zinc-box precipitate. This saving amounts to two-thirds of the present cost of sulphuric acid.

A process for treating the cyanide residue dumps (which are now assuming enormous proportions) is being developed by H. S. Stark. The foundation of this process is the leaching of the dumps *in situ* with sulpho-

cyanate solution and an oxidizer and precipitating the gold from the acidulated solution on scrap iron. The process is now in use at several mines, but few details are available as to the economic results. An application for patent rights on this method has been opposed.

As regards the values of the dumps, it is popularly supposed that the old dumps are far richer (owing to the lower extraction in the past) than those now being made, which contain on an average about 1 dwt. of gold per ton. It must, however, not be lost sight of that, as time goes by, natural leaching of the dumps takes place, any soluble gold discharged with the residue in the weak-cyanide solution adherent as moisture, or as gold subsequently dissolved in the dump by this cyanide solution, being gradually washed out by rain and lost. It is only the presence of encased gold, inaccessible to the cyanide solution, that may then make the dump worth retreating, when regrinding will be necessary.

Of the non-producing mines, there were at work during December, 1905, 35 in the Witwatersrand district, and 22 in the outside districts; giving employment to 3826 whites, 12,583 colored persons, and 4697 Chinese. These mines include several that are rapidly approaching the producing stage. Work in the eastern section of the Witwatersrand has been especially active and mining is now carried on in almost a continuous line for 25 miles east of Johannesburg, as far as the farm Geduld. The main reef has been traced by bore-holes, further east to Palmietkuilen, from which place it takes a turn to the south.

West Africa.—The output of gold in 1905 was 165,844 fine oz. as against 94,815 fine oz. in 1904. This noteworthy increase in gold yield, however, is not reflected in enlarged dividends, the stockholders' returns still remaining incommensurate with the amount of capital invested. The output for the past 12 years, as reported by the West African Chamber of Mines, has been:

GOLD PRODUCTION OF WEST AFRICA.

Year.	FineOz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	FineOz.	Year.	FineOz.	Year.	FineOz.
1894...........	21,331	1896....	23,940	1898....	17,733	1900....	10,557	1902....	29,880	1904..	94,815
1895...........	25,416	1897....	23,555	1899....	14,250	1901....	6,088	1903....	70,763	1905..	165,844

At the end of 1905, the most important producers were, in order of output: Abosso, Ashanti Goldfields Corp., Bibiani Goldfields, Abbontiakoon Block 1, and Akrokerri. New companies in the field are the East Rand Amalgamated Gold, to acquire mineral lands at Tarkwa, Insouta and Bippo in West Africa; the Attasi Mines & Railway, this being a reconstruction of the Attasi Mines, the Attasi & Bokitsi, and the Attasi Goldfields companies; and the Sansu Mine, the last being the reconstructed Ashanti Sansu.

Asia.

India.—Of the total gold yield of India, the Kolar field, in Mysore, furnishes 99 per cent.; the only other producer from lode mining, outside of this small area, is the Hutti mine in Hyderabad, owned by the State.

The output of the gold-producing States is reported as follows:

State.	1900.	1901.	1902.	1903.	1904.	1905.
Mysore (Kolar)	£1,879,085	£1,923,081	£ 1,969,442	£2,283,999	£2,323,183	£2,399,779
Hyderabad (Nizam's)	9,375	....	....	14,505	40,624	50,757
Burma............	3,327	7,606	5,894	3,988	810	(a)
Madras, etc	17	....	....	652	1,462	(a)
Totals (b).....	£1,891,804	£1,930,687	£1,975,336	£2,303,144	£2,366,079	£2,450,536

(a) Not yet reported. (b) Not including small amounts recovered in river washing, for which accurate returns are not obtainable.

The total yield of the Kolar field in 1905 was 627,700 oz. of bullion (564,930 oz.fine), an increase of 4012 oz. over 1904. The five leading mines, together furnishing 98 per cent. of the field's output, are the Champion Reef, Mysore, Ooregum, Nundydroog, and Balaghat, all of which distributed dividends in 1905; in addition Mysore West and Mysore-Wynaad, among those unremunerative in 1904, paid dividends in 1905.

The Champion Reef finds itself in the unaccustomed position of having to combat with poor zones of ore. The mines in the Kolar goldfield have repeatedly had to go through this ordeal. The Mysore mine had this experience at a very early period of its history, but quickly recovered. On the other hand, the Coromandel has spent many years in fruitless exploration for paying orebodies. Ooregum and Nundydroog passed successfully through the trying time at about the middle of their present career. The case of Champion Reef is not immediately acute, for the known ore reserves will last over two years. It is deemed advisable, however, owing to the drop in the values of ore now being developed, to curtail the output, and the amount sent to the mills has been reduced from 15,000 to 12,000 tons a month.

The Champion Reef mine, during the year ending Sept. 30, 1905, crushed 215,167 tons of ore, an increase of 33,219 tons over the preceding year. The year's output of bullion—216,802 oz.—was obtained thus: 188,596 oz. from the batteries; 2617 oz. from No. 2 mill, now dismantled; and 25,589 oz. from the cyanide plant. The cyanide treatment is being curtailed; 177,000 tons of sand was cyanided in 1905. The employment of electric power, secured from a number of independent generating stations, has reduced the mining and milling costs by 4s. 4d. per ton, these two expenses having totaled £1 3s. 3d. ($5.58) per ton milled, in 1905. The average yield has declined from 1 oz. to 0.87 oz. per ton, which was to be expected in the increased tonnage milled. Reserves at the end of the year were estimated at 378,916 tons, a diminution of about 10 per cent. during the

year; during the three previous years, however, reserves had been augmented by 250,000 tons, so that no anxiety is felt from this source. During the year the mine paid £416,000 in dividends on its £260,000 capital.

Gold washing in the rivers is still practiced in India. In the Manbhum district about 300 men are engaged in the industry. Nearly as many men are engaged in washing the sands of rivers in the Central provinces and the Punjab. Gold dredges are successfully operating in the Irrawaddi river in Burma and on the Chindwin river.

Japan.—The gold mining industry in Japan is fostered by the Government, which itself operates the recently discovered veins in the Iwate field. The chief placer deposits are in the Esashi district, Hokkaido. The principal vein-mining centers are in the Echigo, Satsuma and Iwate districts.

Korea.—English, Japanese and American companies are engaged in gold mining in Korea. The largest of these is the Oriental Consolidated Mining Company, of New York. This company operates six mines, five stamp mills and three cyanide plants, and leases three other minès to tributers. During the year ending June 30, 1905, the company mined (including 2826 tons of tribute ore) 257,647 tons of ore, of an average assay of $4.945, at a cost of $0.955 per ton, including all development. This ore yielded, in bullion and from sulphurets, $1,007,715, or $3.91 per ton. Milling of the above output cost $0.57 per ton, in which the cost of cordwood alone came to $0.23 per ton. The five mills comprise 200 stamps; they ran for 320 days in the year, and crushed four tons per stamp per day. The cyanide plants treated 32,556 tons of sulphurets, extracting 75.8 per cent. of their gold, discharging tailing worth $0.78 per ton. The cyanide treatment cost $1.19 per ton handled, the cost of cyanide alone coming to $0.45 per ton. Ore reserves at the company's mines on June 30, 1905, were estimated at 1,068,447 tons, a diminution of 23,355 tons during the year, occasioned entirely by a lack of labor through the war. The Korean Government has been requested by the Japanese to grant permission to Baron Shibusawa and another Japanese to work the gold mine at Hamehung, in Hamgyongdo province.

Malay States.—The results of gold mining in the States during 1904 do not appear to have been brilliant. The quantity obtained from crushings was 12,625 oz. from 54,961 tons in Pahang, and 2189 oz. from 3438 tons in Negri Sembilan. In addition, 146 oz. were obtained from alluvial washings, and 2115 oz. from 11,350 tons of tailings by the cyanide process.

Australasia.

Although the gold mining industry of Australasia showed marked depression during the first half of 1905, its condition improved greatly during the second half; the complete returns for the year, however, show a decrease as compared with the previous year.

The outputs of the individual States for the past three years is compared in the following table:

State.	1903.		1904.		1905.	
	Fine Oz.	Value.	Fine Oz.	Value.	Fine Oz.	Value.
Western Australia	2,064,801	$42,679,437	1,983,230	$40,993,364	1,955,316	$40,416,382
Victoria	767,351	15,861,145	765,596	15,824,869	747,166	15,443,921
Queensland	668,546	13,818,846	639,151	13,211,251	592,620	12,249,157
New South Wales	254,260	5,255,554	269,817	5,577,117	274,267	5,669,099
Tasmania	59,891	1,237,947	65,821	1,360,520	67,897	1,403,431
South Australia	21,195	438,101	29,177	603,089	29,000	599,430
New Zealand	479,715	9,915,709	467,898	9,671,452	492,954	10,189,359
Total	4,315,759	$89,206,739	4,220,690	$87,241,662	4,159,220	$85,970,779

New Guinea.—The Aicora river in the northern territory of New Guinea is the scene of some gold mining, although the unfavorable climate and the difficulty of getting laborers are obstacles. These workings seem to be the richest on the island. Dredging is to be instituted on the Tomato river, where the conditions appear suitable.

New South Wales. (By F. S. Mance.)—The gold won in this State to the end of 1905 is estimated at 12,532,651 oz. fine, valued at £53,235,286. The yield for 1905 was 274,267 oz. fine, valued at £1,165,013, and shows an increase of 4450 oz. over the output for 1904.

As in preceding years the Cobar field has contributed much the largest output of gold, the yield for 1905 amounting to 54,237 oz., a decrease of 7493 oz. fine. The Mount Boppy company still further augmented its output during the year, but the increase from this source was not sufficient to make up the deficiency occasioned by the partial suspension of operations by the Cobar gold mines.

From the Wyalong field an output of 21,813 oz. is recorded, which falls short of the production recorded for the previous year by 4082 oz. As the result, however, of the development work undertaken at the two principal mines the yield for the year 1906 should show a substantial improvement.

Owing largely to the successful operations of the New Hillgrove Proprietary Company, the yield for the Hillgrove division shows an increase of 837 oz., the output for this division for the year amounting to 19,294 oz. fine.

The other divisions which have contributed satisfactory yields are Araluen (14,816 oz.), Wellington (10,684 oz.), Forbes (9542 oz.), Stuart Town (8729 oz.), Adelong (7882 oz.), Orange (6335 oz.), Peak Hill (5830 oz.), Sofala (3807 oz.).

The value of the yield obtained by the gold and tin dredges during the year was £186,994—an increase of £37,158 on that of the previous year. This brings the value of the yield obtained during the last six years up to £690,712. The Araluen division, as hitherto, is the chief center of

gold-dredging operations, the value of the gold won by the 15 plants at work during the year being £62,062.

New Zealand.—The increase in the gold output of New Zealand in 1905 was in striking contrast to the almost uniform decreases of the States in the Commonwealth. While the dredges show a diminishing output, the lode mines, except those in the Coromandel field, increased their yields. The Ohinimuri field, with its two great mines Waihi and Talisman, accounted for bullion worth £931,022; the Thames district, whose leading mine is the Waiotahi, contributed £89,604 and the Coromandel district £9756, giving the total yield of all the Hauraki fields a value of £1,030,382, as compared with £874,562 in 1904. Many of the largest producers have reserves of ore ready for stoping, as a result of recent active development, so that the yield of 1905 will probably be surpassed in subsequent years.

The output from dredges, on the other hand, has shown a constant decline from the record of 1902. Gold dredging in New Zealand, however, is now firmly established as a conservative business, and dividends have not diminished in proportion to the decrease in output. In the Otago and Southland fields, 48 dredges were at work on the average, the total yield from which was 77,222 oz. crude in 1905, as against 89,017 oz. in 1904. On the west coast, an average of 12 dredges gave a total yield of 21,000 oz. crude in 1905, against 26,929 oz. in 1904. The total yield from dredges was thus 98,222 crude oz. in 1905 and 115,945 oz. in 1904.

Queensland.—The decreased output of Queensland is the more noticeable because of the high level sustained for many years; the output in 1905—592,620 oz. fine—is the lowest reported since 1894. Charters Towers, Gympie and Mount Morgan are the leading districts.

The grade of ore raised from Charters Towers, the principal field, was much lower than in previous years, with the result that dividends fell off by fully one-half. This naturally created a depression, which affected the gold mining industry throughout the State. Many of the mines on the Gympie field continue to give good returns, but, although a larger quantity of ore was dealt with, the total output failed to reach that of 1904. The record of the Mount Morgan company is very satisfactory, \$750,000 being paid as dividends during the year. It is expected that the smelting works which are being erected to treat the extensive bodies of sulphide ore will be put in operation early in 1906, and a substantial increase in the output from the mine may therefore be looked for.

South Australia.—Although gold is found over a wide area of South Australia no prominent goldfield has been developed yet. The chief gold-producing center is at Tarcoola, in the central districts, where the Curdnatta, Wilgena Enterprise, Tarcoola Associated and Tarcoola Blocks

mines are prominent producers. The ore averages about 1 oz. per ton and is free milling. It is largely treated in the Government mill and cyanide works.

The Arltunga goldfield also, in Central South Australia, is a comparatively new field and is developing well.

Gold mines are worked in a desultory manner in the Northern Territory. During 1905 promising placer finds were made between the Daly and Fitzmaurice rivers and at Winnecke.

Tasmania.—Although the gold yield of Tasmania is small in comparison with that of the other States still it continues to show a steady advance. At the principal mines, the Tasmania and the New Golden Gate, additions have been made to the plants, and the mines have been placed in a position to augment their output. At the Tasmania mine, a pumping plant capable of lifting 8,000,000 gal. of water per day from a depth of 2000 ft. has been installed. The gold obtained by the Mount Lyell company in its blister copper also goes to swell the total yield of this State.

Victoria.—Among the Eastern States, Victoria still holds its prestige as the chief producer of gold. The bulk of the yield was contributed by the Bendigo field, and, although the output was less by about 33,000 oz. than that of 1904—which was the highest for the previous 30 years—still it was such as to furnish unmistakable evidence of the richness and permanence of the reefs. In the New Chum Railway and the Victoria Quartz mines, the existence of reefs carrying gold at the depths of 4230 ft. and 4090 ft., respectively, was demonstrated during the year, and this still further strengthened the confidence of investors in this field. The aggregate returns from the Ballarat district vary little from those of the preceding year. The yields from the mines at Walhalla and Berringa show that the industry has made headway in these centers. From many of the mines working the deep leads at Rutherglen, Chiltern, Maryborough, and Creswick, good returns were reported. The work of draining the deep and wet leads is being vigorously proceeded with, and substantial progress has been made, but the undertaking is not a light one, as instanced by the Loddon Valley mine, where 12,000,000 gal. of water has to be lifted daily (8300 gal. per min.). Increased yields are recorded as the result of the operations of the various dredges, but ordinary alluvial mining exhibits a decrease.

Western Australia. (By H. C. Hoover.)—The gold production of this State during 1905 was again marked by a steady decrease; in total about 28,000 oz. less than for 1904, and about 110,000 oz. less than 1903. There is every reason to anticipate a further decline in future, barring, of course, the discovery of new districts. The decline is due, in part, to each of the following causes: Decrease in value with depth in the leading mines; lower working costs, and therefore lower grade ore included in

mine products; exhaustion of surface alluvial; and failure in the discovery of important new mines.

The first and last are the most serious. The decrease in value with depth

GOLD PRODUCTION OF WESTERN AUSTRALIA.

	1902.	1903.	1904.	1905.
Jan.	144,496	178,360	176,653	165,452
Feb.	131,206	163,058	162,402	154,033
Mar.	152,527	165,031	136,835	160,918
April	157,705	176,814	180,999	172,136
May	141,117	175,859	160,280	157,685
June	102,938	176,333	167,446	155,149
July	158,304	180,097	160,719	166,006
Aug.	161,521	175,110	168,432	174,681
Sept.	161,689	170,693	165,852	163,297
Oct.	167,034	161,692	174,607	160,622
Nov.	169,448	170,479	163,933	163,248
Dec.	163,053	171,257	165,072	162,090
Totals	1,871,038	2,064,801	1,983,230	1,955,316

cannot be helped; but the failure to develop new mines of consequence during the last seven or eight years is due to a large decrease in prospecting. The conditions of the Australian Mining Title, by which a leasehold is granted subject to maintenance (the constant employment of one man for every six acres), have rendered it practically impossible for the prospector to hold his ground; and, although personal concessions are constantly given by officials, the fear of inability to hold ground has driven the prospector largely from the field and throttled the growth of the industry. Politically the Colony has freed itself from a labor government, and more can now be hoped toward the development of the State's resources.

The dividends for the year I estimate at £2,150,000. Several companies have strengthened their reserve funds to provide a war-chest for a campaign to find another mine to take the place of present properties on exhaustion. Several are looking beyond the State of their birth for such opportunities. The nominal capital of all consequential West Australian companies quoted on various markets, which embraces practically all dividend-paying concerns, is at present about £5,320,000. The value of these companies on the market is £15,670,000; showing that the average return for the year was roughly 40.41 per cent. on the nominal capital, and 13.72 per cent. on the market value of the shares.

The depth now reached by the leading mines varies from 1000 to 2000 ft. There has been an undoubted decrease in value with depth; and especially Kalgoorlie appears to be a good field for the confirmation of theories as to secondary enrichment of the sulphide zone. The decrease in average of output per ton has not been due to this cause alone, however, for the great decrease in working cost, with improved extractions, has permitted the inclusion of large amounts of ore hitherto unprofitable.

The total ore reserves in the leading mines which furnished fully 70 per

cent. of the State's output are about 5,000,000 tons. The annual extraction is now at the rate of about 2,000,000 tons; so that somewhat over a two years' supply is visible in average. Individual mines vary from a few months ahead of the mill to as much as five years.

Working costs have shown a further improvement during the year. The wide variables as to size of orebodies, degree of refractory character, distance from rail, size of equipment, etc., prevent any generalization of figures. The following are the present average costs for all charges, except development, on several typical mines. These amounts include administration, repairs, renewals, etc. The cost of development varies on these mines from 30c. to $1 per ton.

	Great Fingall, 100 Stamps, 18,000 tons per month.	Ivanhoe, 100 Stamps, 17,000 tons per month.	South Kalgurli, Dry Crushing, 7,000 tons per month.
Mining, per ton	$1.68	$2.04	$1.62
Treatment, per ton	1.94	2.14	2.78
Realization, per ton	0.08	0.10	0.04
General expense, per ton	0.08	0.12	0.28
	3.78	4.40	4.72

No startling metallurgical advance was made during the year, although all departments show steady improvement. The average of Kalgoorlie extraction on well-managed mines is probably over 90 per cent. on telluride ores; one mine, the South Kalgurli, is averaging over 94 per cent. The most important advance has been made in the elaboration of regrinding as an adjunct to the increase of stamp duty. The question of regrinding was first undertaken on Kalgoorlie mines, as very fine division, amounting to absolute slime in most instances, is a necessity in securing efficient extraction. Various machines were introduced for this purpose, all forms, such as Huntington mills, etc., being discarded in favor of an adaptation of the old Wheeler pan, and the tube- or flint-mill. Which of these latter machines will ultimately prevail is yet disputed, but sufficient experience has been gained to settle some features of practice.

The pan is the best machine yet found for grinding sand from one degree of coarseness to another; and where regrinding is installed as simply an adjunct to increased mill-duty (in crushing sand from, say, 10 or 12-mesh mill-screen to about 30 mesh for percolation-cyanide treatment), it is superior to any invention yet brought forward. To W. J. Loring belongs most of the credit for advancement along this line. The Sons of Gwalia, Great Fingall, and Oroya mines, under his general charge, received most of the investigations. The Sons of Gwalia 50-stamp mill is now crushing 11,000 tons per month, and further installation now in progress will increase it to 12,000 tons. The Great Fingall mill, of 100 stamps and on

very hard quartz, is doing more than 18,000 tons per month, and will probably do 20,000 tons. On these mines, sliming is undesirable, and the increase of slime over a similar degree of fineness from mill-screen is not over 7 per cent.

Where sliming is necessary to increase extraction, the relative merits of the tube-mill and pan are still unsettled, both having strong advocates. The Ivanhoe mine, under R. B. Nicolson, has done the most efficient work, and is the leading exponent of the pan, as the Oroya-Brownhill is the leading exponent of the combined pan and tube-mill. At the Oroya, 10,000 tons per month are crushed by 50 stamps, and at Ivanhoe 17,000 tons with 100 stamps. At the latter mine, tube-mills have been discarded, but the mines are not quite on a parallel basis, as a portion of the product of the Ivanhoe mill is treated by percolation, and it is difficult to compare with the Oroya, where all the ore must be reduced to a slime. In the latter case, pans are used for intermediate crushings between the stamps and the tube-mills.

In general, it is certainly settled as to the tube-mill that it is a sliming machine; and where an increase in slime is undesirable, it has no place at all. Equally well is it settled that the pan is the superior machine for crushing sand to an intermediate stage. It is yet to be determined as to which machine will prevail where sliming is desirable.

Europe.

Great Britain.—The only gold mine in Great Britain, that belonging to the St. Davids Gold Mines, Ltd., continues to yield profits. It is well known that the gold deposits in Wales are variable in value, and mines have often been taken up and abandoned when the first barren patch had been struck. The St. Davids company has now been in existence for seven years with fair success. In 1905 it produced 15,538 tons of ore yielding 5551 oz. bullion, equivalent to 4831 oz. fine gold. After spending £5140 for new development and £862 for royalties, a surplus of £2841 was left. A third lode has been opened and promises well.

North America.

Canada.—The total gold production of the Dominion in 1905 showed a decrease of nearly \$2,000,000. from the year before. The largest contributor, as for several years past, was the Yukon Territory, which is to be credited with over 57 per cent.; British Columbia with nearly 40 per cent.; Nova Scotia and Ontario, together with a little from Quebec and Alberta, 3 per cent.

In Nova Scotia there was a slight recovery from the general decrease which has been apparent for a number of years. The explanation would seem to be that the immediately accessible surface ores of most of the districts have been largely worked out, and the revival of the industry

will depend upon the inauguration of a radically different policy from that which has been followed so far. It is believed that consolidation of many of the numerous smaller mines, and the inauguration of new methods and plant suitable for the economical mining of ore from greater depths, will lead to renewed activity. As these matters seem to be receiving serious attention, a revival of the industry is looked for.

In Ontario, activity was evident in prospecting and developing at numerous points both in the old districts in the eastern parts of the province and in the newer gold-bearing districts west of Thunder Bay.

In British Columbia, a dry season is said to have affected the production of placer gold, but as a whole the industry was about as in 1905.

The output of the Yukon Territory placers continued to fall off, as it was expected they would, and this will probably continue until under favorable conditions the more permanent forms of the lower-grade bodies of gravels are fully established.

For a considerable time the production of silver has been nearly altogether accounted for by the silver contents of the various ores of other metals mined and treated in British Columbia. This province furnished 90 per cent. of the metal during 1905. Owing, however, to the discovery and working of the exceedingly rich silver-cobalt-nickel ores near the northern end of Lake Timiskaming in Ontario, that province has suddenly attained great importance.

The shipments from Cobalt, Ont., have been to smelters, chiefly in the United States. The results apparently have not been satisfactory to the shippers, and efforts are being made toward the erection of smelters to treat the ores locally. At Copper Cliff, the Canadian Copper Company has already erected a plant capable of partially treating these ores.

The silver occurs mostly in the metallic condition and although the veins are small, being measured in inches of thickness instead of feet, the silver contents are so high that, although only in the second year of its existence, the aggregate value of the shipments has been extraordinary, especially in view of the comparatively limited amount of the development work done. Carloads of ore valued at $60,000 to $100,000 have not been unusual. For the first six months of 1905, official figures give the shipments as 891 tons, aggregating $688,004, or a little over $772 per ton. These were from six mines, the existence of which dated only from the previous autumn.

A recent bulletin, issued by the Canadian Geological Survey, draws attention to another important discovery on the west side of Windy Arm, a southerly branch of Lake Tagish. This is situated in the southwest part of the Yukon Territory and is easily reached from the White Pass & Yukon railway. The report describes the veins as strong, persistent, and already traced for considerable distances. The widths are of from

1 to 5 ft. in one case, and 9 ft. in another. The main value of the ore is in silver and gold. Antimonial and arsenical silver minerals seem to be the most common form of that metal.

British Columbia. (By E. Jacobs.)—The dry season was accountable for the decrease in placer gold. Various estimates published in provincial newspapers have credited Atlin with an increase of $100,000 and upward, but these have not been confirmed from official sources. Cariboo output was smaller, owing to the unusually short run on the Consolidated Cariboo Hydraulic Company's big mine, at which the recovery, by reason of a shortage of water, was by far the smallest in nine years. The increase in lode gold was contributed largely by the copper-gold mines of the Boundary district and the Nickel Plate mine near Hedley, Similkameen. Rossland, however, showed a small decrease, thereby reducing the net increase.

The St. Eugene mine, at Moyie, East Kootenay, contributed the larger part of the increase in silver, and the Sullivan Group mines, near Marysville, East Kootenay, added a share. Boundary also showed an increase, but the Lardeau, in West Kootenay, and the Coast districts produced less than in 1904.

GOLD AND SILVER PRODUCTION OF BRITISH COLUMBIA.

	1902.		1903.		1904.		1905.	
	Oz.	Value. (a)	Oz.	Value. (a)	Oz.	Value. (a)	Oz.	Value. (a)
Gold, placer....	53,657	$1,073,140	53,021	$1,060,420	55,765	$1,115,300	48,465	$969,300
Gold, lode.....	236,491	4,888,269	232,831	4,812,616	222,042	4,589,608	238,660	4,933,102
Total gold.	290,148	$5,961,409	285,852	$5,873,036	277,807	$5,704,908	287,125	$5,902,402
Silver.........	3,917,917	1,941,328	2,996,204	1,521,472	3,222,481	1,719,516	3,439,417	1,971,818

(a) Placer gold is valued at $20 per oz.; lode gold at $20.67 per oz.; silver at average market quotations.

Nova Scotia.—The gold mines of Nova Scotia center within a short radius of Halifax; the largest output was recorded in 1901, since which year the annual yield has relapsed to its dimensions of 40 years ago. The yield during the year ending Sept. 30, 1905, was 15,549 oz. as compared with 14,270 oz. in the previous year. This does not include the gold contents of the antimony ore from the Dominion Antimony Company's mine at West Gore. During 1905, this mine produced 4000 tons of ore, which was sorted into two classes: 430 tons of the first-class ore carried 46 per cent. antimony and 2.56 oz. gold per ton; 3570 tons of second-class ore carried 8 per cent. antimony and $10 in gold per ton. Of this year's output, 527 tons, carrying 1233 oz. gold, were shipped to English smelters. The owners of the mine now receive the value of only half of the gold, but large-scale experiments are now under way, under the advice of MacArthur and Forest, directed toward the recovery of both antimony and gold, at the mine. The proposed treatment has been described, under "Antimony," on an earlier page of this volume.

The gold mines of Nova Scotia, in 1905, were the subject of professional

investigation by T. A. Rickard, on behalf of the Provincial Government. His report is not yet made public.[1]

Yukon. (By J. P. Hutchins.)—The gold production of the Yukon in 1905 was about $7,000,000, or 30 per cent. less than in 1904. The Klondike is a placer district, no paying quartz mines being in operation, and it has now passed its zenith. Last year was in the main unfavorable for large production. There was a light snowfall, abnormally late, and the resulting condition of the winter trails interfered with hauling wood to the claims. The rapid thaw was not conducive to best results in sluicing the winter dumps. However, little loss resulted by freshets washing away material dumped in the creek bottoms or by flooding open cuts.

The stampede to Tanana during 1904 included many of the men who usually winter in Klondike, and work "lays" for want of something better to do. Considerable low-grade ground (unworkable during the summer by reason of thawing, and not profitable unless the lower scale of winter wages—30c. per hour and board—is in force) is ordinarily mined. Last winter few "laymen" mined, and part of the decrease is attributable to this cause. Lay percentage, that is, the share to the "laymen," was higher and varied from 60 to 80 per cent. of the total output.

The mining population numbered only about 3000; six years ago it was more than 20,000. In a normal year labor would have been scarce, but the dry spring and summer made fewer miners requisite. Labor was scarce during the fall, when heavy rains furnished ample water for mining purposes. The rapid thaw and dry spring prevented ground sluicing of "muck" and other overburden from creek, hillside and bench claims for work during the summer and fall.

The advent of cold weather, and the cessation of mining operations, were normal. Less ground was reserved for exploitation by the hydraulic method, the extreme popularity of this means of working alluvion having been followed by a slight reaction; the production was not materially affected in this way.

The Canadian Yukon includes not only the Klondike proper, but several other camps, including Stewart river, Atlin, etc. Klondike is the main producer. The following tabulation shows the gold production of Yukon territory, the total to date being $113,175,000:

Year.	Amount.	Year.	Amount	Year.	Amount.	Year.	Amount.	Year.	Amount.
1896......	$ 300,000	1898......	10,000,000	1900......	22,275,000	1902......	14,500,000	1904......	10,350,000
1897......	2,500,000	1899......	16,000,000	1901......	18,000,000	1903......	12,250,000	1905......	7,000,000

A remarkable variety of methods for mining the placers of the Klondike has always been noticeable. During the early days, the practice was as

[1] A description of the more general features of the district was published in the *Mining and Scientific Press*, Nov.-Dec., 1905.

varied as nationality. There were few experienced placer miners, and they were not familiar with handling frozen material. Some devices of startling ingenuity were used. Surviving devices, the offspring of some of the numerous mechanical mistakes, are still in existence. In the methods of mining, due particularly to the frozen condition of alluvion, there is a generally accepted system, but in stripping off overburden and in transporting pay dirt to sluices, there is a striking diversity of practice. Overburden in creek bottoms is stripped by ground sluicing, steam scraping, horse scraping, steam shoveling or hand shoveling. Steam scraping (a method possessing many advantages and which may be well used in other placer districts) is still the favorite method.[1]

Larger and more powerful scrapers, engines and boilers were used in this work, and with better results during 1905. The means of transporting pay material to the sluices were more varied than ever. Shoveling to platforms, thence to sluices; shoveling into wheelbarrows, wheeling to bucket, raising on inclined cableway to sluices; steam shoveling direct into sluices; shoveling into skips, hoisting by derrick to sluices; shoveling into cars, hauling on inclined track to sluices, were the most common. Shoveling into wheelbarrows, wheeling to bucket and hoisting on inclined cableway is still the most generally practiced method of transporting pay-dirt.

One innovation was the use of a steam shovel to load a skip, in conjunction with a Lidgerwood cableway system, to transport material to a trommel and to gold-saving devices. This arrangement has won numerous advocates, who claim for it many advantages in ground which is not workable by the floating dredge. Any definite conclusion regarding a superiority of this method over that including steam scraping as a way of transporting material at present must be considered premature. This is particularly true of frozen area; the removal of thawed material in successive layers is best accomplished by scraping.

Three dredges were installed during the year and results were satisfactory. The expansion in dredging operation is due to a more thorough appreciation of the advantages of this method when applied to areas thawed, or but slightly frozen, and to a more general realization of its ability to excavate sufficiently most of the schist and shale bottoms.

A unique installation was made on the Klondike river. An electric dredge, with 7-ft. close-connected buckets, was built and made ready for operation in 43 days. A steam turbine was used to generate electric power with the object of economizing in fuel and permitting more deck-room. This took care of sudden loads of 100 h.p. without difficulty, and was found to be entirely satisfactory. The ordinary steam dredge is wasteful of fuel and has little deck-room.

In open-cutting in frozen ground (except where mechanical excavators

[1] "Methods and Costs of Placer Mining in Alaska," U. S. Geol. Survey, *Bulletin* No. 263, p. 61.

such as steam shovels and dredges are used) little thawing is required, the overburden being scraped and the "pay-dirt" handled as it thaws by exposure to sun and air. No new device for thawing frozen alluvion has been used. A more general realization of the futility of attempting to handle material when frozen was noticeable. It is possible to excavate frozen gravel if large and powerful steam shovels with small dippers are used; but the impossibility of a satisfactory gold extraction from gravel which is not completely thawed makes such procedure bad practice. Thawing cost has not varied materially; when steam is used on an economical scale, an allowance of about 40c. per cubic yard is made.

The year was not favorable for hydraulic mining; the early, warm and dry spring, the rapid thaw, and the dry summer caused a shortage of water. The copious fall rains were too late to be of the best advantage. Yet in spite of the bad season about 1,500,000 cu. yd. (50 per cent. more than in 1904) was washed. This was due to the conservation of water in reservoirs, and the utilization of more of the flood-water, formerly wasted during the spring thaw, in installations completed during 1904. There was no material modification of hydraulic mining methods. There was less ditch construction than in 1904; the largest ditch excavated in 1905 is 7 miles long, with a capacity of 10 sec. ft. A dam, to be 50 ft. high, forming a reservoir with capacity of 60,000,000 gal., was begun. The significance of these figures is patent as demonstrating the lack of favorable reservoir sites in Klondike.

No great difficulty was experienced with dams built during 1904, although some was anticipated. All earthwork is hampered by the frozen condition of material. Cost of hydraulicking, including amortization, etc., has not fluctuated greatly; 20c. per cu. yd. is proper for well managed operation. A duty of 5 cu. yd. per miners' inch was attained in several authentic instances. Considerable difficulty, due to conflicting water grants, issued under faulty mining regulation, was experienced during 1905. Few applications and fewer grants were recorded in 1905, those of previous years having covered practically all available water. Hydraulicking with pumped water was still carried on by the same operators with results similar to those of 1904. This method is very costly and is seldom profitable.

The physiography of the Klondike is such that an ample and constant supply of water is not attainable without long ditches, flumes and inverted siphons. Climatic conditions are extremely adverse to the construction and maintenance of such water ways. In addition to the high cost due to expensive labor, materials and transportation, and the hostile climate, good ditching or fluming ground is seldom encountered, and maintenance expense is extremely large. Water has been obtained from limited areas, topographic isolation preventing other procedure. This circum-

stance has been a tremendous handicap, making slow and costly methods necessary, where an ample water supply would have allowed rapid and cheap exploitation. During 1905, the Canadian Government had a party in the field investigating the possibility of obtaining a constant supply. No actual ditching to bring a large supply of water from a considerable distance has yet been undertaken.

There were no developments of new or rich alluvial deposits during 1905. The genealogy of the placer deposits and their simplicity of occurrence are such that no discoveries of "deep" or "back" channels is likely. The existence of peneplains of the "White Channel" occurring as secondary benches and terraces, though none have been found, is likely. This "White Channel" is what is left of the ancient creek-beds; it has a course approximately parallel to, and an elevation of 150 to 300 ft. above, the present creeks. Where it has not been eroded it appears as a bench deposit with one rim completely removed. There are no paying quartz mines in Klondike, though some prospecting for payable veins was carried on in 1905. The notion of a "mother lode" still attracts many prospectors whose conception of geologic history and appreciation of the colossal erosion is deficient. The wonderfully concentrated deposits of Klondike (where lateral and vertical dissemination is so seldom found) are the results of the concentration of prodigious volumes of what was probably low-grade material.

The development of more so-called "worked out" ground during 1905 was gratifying. Early methods contemplated working only the richer spots and were crude and wasteful; with improved exploitation, much ground formerly thought to be "worked out" has become valuable. No unusually rich ground, such as was common in the early days, was mined.

As noted above, the supply of labor was not ample. Wages are still high—40c. per hour and board in summer, 30c. per hour and board in winter being the ruling rates. There are no unions and no strikes have occurred. An unusually high class of labor is available, the tremendously stimulating climate having developed a remarkable body of men.

Wood hauled on runners cost from $10 to $12.50 per cord; for that delivered on wheels $14 to $22 per cord was paid. It was usually delivered in 16 ft. lengths. The cost of fuel in Klondike is materially affected by length of haul; the rapid denudation of adjacent hills, besides preventing conservation of water, has caused a yearly increase in fuel expense. Experience with local coal has not been encouraging; during 1904 it was found to be of low calorific value, and it is only used where peculiar conditions make its cost per ton less than that of wood per cord. A ton of coal has been found to be equal to a cord of wood in steam-producing qualities. A line of river steamers used about 6000 tons of

coal from a mine about 200 miles from Dawson, on the upper Yukon river.

New wagon roads and trails have been built in Stewart river and Miller creek districts. The excellent roads already constructed have been well maintained; large loads (20 cwt. per animal) are hauled on them. Alaska mining camps, still notoriously deficient in this respect, can profit by this example. The construction of a narrow-gage steam railway from Dawson to Grand Forks, 12 miles distant, has been begun. Transportation cost for supplies from the "outside" (southern ports of British Columbia and northern ports of the United States) has not varied materially; about \$60 per ton is charged. Passenger rates for railway, river-steamer and stage travel are about as formerly, 20c., 11c., and 25c. per mile being the respective average charges.

The law still requires \$200 worth of work per annum on each claim or fractional claim; this is called "representation work." Title cannot be obtained, but claims are leased by the Crown to licensed "free miners" from year to year, as long as the rules are followed. The regulation passed in 1904, to allow holders of water rights to sell surplus water because of the dry season, was of little benefit during 1905. What little water was sold commanded the high price of \$1 per ten-hour miners' inch. No new regulations as to concessions were passed. No new concessions were granted, and little work was done on those already existing.

Mexico.—This country leads all others in the production of silver, the mining of which metal extends back to remote antiquity, and in the production of gold has advanced to sixth place, having already surpassed India and become a close competitor of Canada. Silver is yielded by every mining State, but the richest gold fields are at El Oro, in the western part of the State of Mexico, where the principal mines are those of the El Oro Mining and Railway Company, Esperanza Mining Company and the Dos Estrellas.

EXPORTS OF GOLD AND SILVER FROM MEXICO.
(Fiscal years ending June 30. Values in United States currency.)

Gold.	1903–04.	1904–05.	Silver.	1903–04.	1904–05.
Mexican coin	\$ 53,834	\$ 42,955	Mexican coin	\$ 9,335,798	\$ 949,946
Foreign coin	1,463	19,561	Foreign coin	43,499	38,985
Bars	5,042,329	6,264,313	Bars	22,894,053	26,507,008
All other forms	265,594	521,244	All other forms	7,281,995	5,265,883
Total gold	\$5,363,220	\$6,848,073	Total silver	\$39,555,345	\$32,761,822

The report of El Oro Mining and Railway Company, Ltd., for the year ending June 30, 1905, shows reserves of ore definitely known at the end of the year to be 672,850 tons, with an average value of \$9.30 in gold, besides 3 oz. silver. Reserves of the high-grade ore are diminishing, while new openings are revealing mainly ore of lower value; thus, in the "hanging-wall orebody," and in the "branch vein," whose gold content averages

$14.70, only 97,800 tons remained at the end of the year, the rest of the reserve averaging $8.30 in gold to the ton.

Improved resources are anticipated in the Somera No. 1 claim, lying to the west of the existing mine, on which development has been begun within the year. A crosscut into this ground from the Somera shaft has already cut three new veins, and others, already developed on other properties adjoining on opposite sides of the Somera claim, are confidently looked for.

Of the 142,181 tons of ore broken and hoisted from the mine, the "hanging-wall"and the "branch" vein, the two higher-grade ($14.70) orebodies, gave 104,104 tons, the remainder consisting of low-grade ($8.30) ore from other lodes. The stopes on the 186-ft. and 286-ft. levels, in the "hanging-wall" vein, are exhausted.

The company has two stamp and cyanide mills, the second of which came into full operation in May, 1905, its construction having cost $746,537 within the last two years. The new mill has 100 stamps, and is making a trial of three tube-mills. Two-thirds of the ore is reduced to slime, and the remaining sand is ground to between 100- and 200-mesh. An improved extraction of both gold and silver is observed as a result of this practice. The older mill ran throughout the year, its slime having been diverted to the tanks of the new mill as soon as these were ready. Milling operations may be summarized thus:

Details.	No. 1.	No. 2.	Percentage of extraction:	No. 1.	No. 2.
Days run	348.5	71.7	In mill— Gold	12.64%	6.49%
Tons crushed	122,006	20,565	Silver	1.75	3.19
Tons per stamp per 24 hrs.	3.34	2.26	By cyanide—Gold	66.00	70.24
Assay value of ore:			Silver	52.50	56.45
Gold	$12.91	$9.16	Total gold	78.64	76.73
Silver	2.50	1.80	" silver	54.20	59.64

The total value of the ore treated was $2,069,684 and of the bullion recovered, $1,527,723; additional bullion valued at $16,114 was recovered from slag, old plates, and concentrate.

The apparent inferiority of the new mill in saving gold, the average being for the first three months of its campaign, does not exist at present. In June, the last month recorded, it was saving 80.3 per cent. of its gold.

Costs of mining and development increased, but the expenses of milling and cyaniding decreased during the year. The costs per ton, on the basis of 142,571 tons mined, were as follows: Mining, $2; development, $1.11; milling, $1; cyaniding, $1.21; miscellaneous operating, $0.25; general, $1.27; water supply, $0.03; total, $6.87.

Electric power has been contracted for with the Mexican Light and Power Company. This is expected to reduce working costs to $5 per ton and to repay the cost of installation in two years; it was to be in operation in October.

(By James W. Malcolmson.)—The greatest advance during 1905 was probably in the gradual and general replacement of antique methods of mining and ore treatment by modern machinery and modern processes. This is especially noticeable in the older districts, such as Pachuca and Guanajuato. These camps have produced no small fraction of the total silver supply of the world; they have been in operation continuously for centuries; they are taking on new life. In Pachuca the ancient patio process is retained, but the ore is concentrated over Wilfley and Johnson tables before amalgamation, and the horses have been replaced by plows operated by electric motors. In Guanajuato, after preliminary concentration, the cyanide process, in a modified form, has displaced the patio process entirely, on account of the higher gold values in the ores. In both camps electric power is displacing the old methods of hoisting and pumping.

Guanajuato.—In this State the property of the Guanajuato Consolidated Mining and Milling Company is now on an excellent footing. The mine is well equipped; large reserves of good ore have been discovered and opened up in virgin ground; the concentration and cyanide plants are giving good results, and the company looks with confidence to a satisfactory future. Elsewhere in Guanajuato mines are in process of development, mills are being erected, and in one or two instances, notably in the property of the Peregrina Mining and Milling Company, there is enough ore developed to assure a substantial profit on operations. This is largely owing to the gold which was not amenable to older processes. Several properties now being placed on the market, however, are not in this condition; before they can be put upon a profit-paying basis, new orebodies must be found, or the extensions of older deposits must be opened up.

Hidalgo.—The mines of Pachuca and Real del Monte, which are largely owned and operated by local capitalists, were extremely prosperous in 1905. The Real del Monte Company, controlling the Barron, Dificultad and Camelia mines, practically reconstructed its mill at Loreto; the San Rafael and Santa Gertrudis companies sunk their main shafts and opened up good ore on the deepest levels. Three-thousand horse-power is transmitted to this camp from Regla and from the overflow of the drainage of Mexico City. The La Blanca mine reduced its output, on account of difficulty in its main shaft; but exploration work was satisfactory.

Mexico.—The richest orebody in Mexico for several years has been opened up by the Esperanza Mining Company, operating in El Oro, in the State of Mexico. Late in 1904 a new vein was cut by diamond drilling, east of the older workings; by the latter part of 1904 monthly profits of $350,000, United States, were earned. The value of the gold and silver mined was $800,000, United States, monthly. During 1905 enough ore was discovered and opened up in the Esperanza mine to leave a total net profit of over $10,000,000, United States currency. Of the other companies operating

in this district, the Dos Estrellas and the El Oro Mining and Railway Company maintained their production and increased their mill capacity during the year. The most promising unexplored area in this camp is that lying between the El Oro and Dos Estrellas mines; this is covered by the Somera and Victoria claims. Exploration in this ground has commenced on a large scale.

Sonora.—At the Cerro Prieto property, of the Black Mountain Mining Company, in Sonora, a large deposit of gold ore has been opened; a plant is being erected at Magdalena, on the Sonora railroad. Water-tube boilers, steam turbines and electric generators will produce power at Magdalena which will be transmitted to the mines, 25 miles away, where the crushers, stamps, air-compressors and pumps will be operated, at an estimated cost of $100 United States, per horse-power year, coal costing at the boilers $6.50, American currency, per ton. One hundred stamps have been installed, and ore will be treated by amalgamation and cyanidation.

The Greene Consolidated, of Cananea, is a large producer of gold and silver in this State.

Chihuahua.—The Batopilas Mining Company uncovered a bonanza of native silver, which was worked throughout the year. By the end of October, 2,000,000 oz. of silver had been mined. This mine is six days by horseback from the railroad, and has been developed with great expense and difficulty. A tunnel 9000 ft. long has been driven to open the veins; the mills are operated by water-power. The Batopilas silver mines have been worked since the 17th century, and made larger profits in 1905 than during any previous year.

The production of ores from Santa Eulalia increased materially in 1905, the principal producers being the Potosi Mining Company, the Chihuahua Mining Company, and the Santa Eulalia Exploration Company. This district is now the principal mining center of Chihuahua. The completion of the new smelter of the American Smelting and Refining Company, at Chihuahua, 16 miles away, will enable lower-grade ores to be handled with profit. At present the camp is distant from the nearest smelters at El Paso and Torreon, 240 miles and 310 miles, respectively; this constitutes a serious drawback. The building of the Kansas City, Mexico & Orient railway, which is now in operation 72 miles east and 185 miles west of Chihuahua, has made that point a natural smelting center.

Customs Smelting.—On account of the unsatisfactory condition of Mexican lead ore production, suitable for customs smelting, the increasing production of gold and silver ores, most of which is silicious, is seriously handicapped. It has become evident that the customs smelters using lead as a collector have not sufficient capacity to handle the output of gold and silver ore, and treatment rates on silicious ores have risen throughout

the country. The result is that mines formerly operated with profit have been compelled to shut down, and the silver output of Mexico is materially reduced. This condition was made worse by the sudden increase of silicious ores mined and shipped to the smelters from El Oro during 1905.

Copper as a collector is being more largely used in customs smelting than formerly, and it appears certain that the new plants now being started at Chihuahua, Velardena, Oaxaca, Angangueo and Terrazas will use copper to a larger extent than at the older customs smelters. The Metallurgical company, of Torreon, Coahuila, has also decided to make an addition to its plant for the purpose of smelting silver and gold ores with copper.

Nicaragua.—Mining in the Piz Piz district is active, and work is in progress on a number of properties. The Bonanza mine is producing 200 tons daily of good grade free-milling ore, all the ore coming from open cuts. Eight Huntington mills reduce the ore. A large slimes plant has been added to the cyanide plant.

South America.

Argentina.—Several companies have been organized to undertake dredging in Argentina and Patagonia. There are believed to be some extensive placers along the base of the Andes, and in lower Patagonia and Tierra del Fuego.

Bolivia.—The Compania Minera San Juan de Oro is operating a gold dredge at Tupiza, near the Argentina boundary. The auriferous deposits of eastern Bolivia are well known, though but little exploited. With this beginning and with pay ground to work, the problem of tracing out the extensions southerly will receive attention.

Engineers representing prominent English interests examined the celebrated Arque mines, between Oruro and Cochabamba; their examination is expected to result in the development of the properties.

Brazil.—The exports of gold bullion during the last four years have been: 3990 kg. in 1902; 4322 kg. in 1903; 3871 kg. in 1904; and 3040 kg. in the first nine months of 1905.

During 1905 the Ouro Preto Gold Mines crushed 75,660 tons of ore valued at $5 per ton. It is estimated that there are 208,000 tons of ore at present in sight in the workings, which is of higher grade than the ore treated during 1905. The average working cost amounted to $5 (U. S.) per ton.

Colombia.—The three principal departments productive of precious metals at present are Tolima, 600 miles, and Antioquia, 450 miles from the Atlantic coast, and the Cauca, which stretches from the Pacific coast landward.

The gold veins in the Tolima district with few exceptions are rich superficially, but pinch out at a depth of 10 to 20 fathoms. Alluvial gold washings are more abundant and give better results. Free gold quartz veins are the most worked in Tolima, as no reduction works have been set up to

treat compound minerals, although many lodes of this nature exist. Silver mines are abundant in this department, and several mines export their concentrated minerals to Swansea, Wales.

Antioquia is the most active goldfield of Colombia. Considerable gold is exported annually, the greater part being obtained from alluvial gravel by pan washing by the natives, but valuable gold mines are being worked to advantage at deep levels. The district of Remedios is rich in lodes carrying refractory ore.

The Cauca department is most promising, but is the least explored. It is rich in many minerals, and the few mines at work are successful.

Guiana, British.—The output of gold in 1905 was 95,253 oz., crude, as against 90,709 oz., crude, in 1904. The 1905 yield was equivalent to 82,300 oz. fine gold, valued at $1,701,141. Exports in 1905 were valued at $1,637,685 as against $1,599,043 in 1904.

Guiana, Dutch.—The output of gold in 1905 was the largest ever recorded, and is noteworthy from the fact that all of the gold produced in the Colony has been by hand-work using crude long toms and sluices.

GOLD PRODUCTION OF DUTCH GUIANA.

Year.	Surinam.	Saramaca.	Murowyn.	Lawa.	Total.
	Kg.	Kg.	Kg.	Kg.	Kg.
1896	418.0	151.3	65.9	191.2	846.4
1897	434.7	135.5	83.0	250.0	903.1
1898	441.1	120.9	78.4	224.5	865.0
1899	400.8	131.6	70.1	290.6	893.2
1900	359.3	150.4	66.8	299.5	876.1
1901	254.7	119.1	9.8	369.2	752.8
1902	231.1	88.7	17.8	249.1	586.6
1903	243.2	149.5	56.0	233.7	682.5
1904	294.2	242.8	133.5	241.3	801.8
1905	416.1	213.5	188.5	253.1	1071.3

A large part of this has been by what is known as the á la par or "pork-knocking" system, whereby the concessionaire receives from 10 to 20 per cent. royalty on all gold produced. At present a dredge is being installed by an American company under the direct supervision of an experienced engineer, who has spent time enough on the ground to satisfy himself as to values, climatic and labor conditions. Records of the Paramaribo office show duty paid for 1905 on 1071 kg. produced from the district as follows: Surinam, 416; Saramaca, 213; Murowyn, 189; Lawa, 253 kg.

Progress in Gold-Ore Treatment During 1905.

By Alfred James.

The main features in gold-ore treatment during this year have still been finer crushing, and the treatment of slimed products. From Australia and Africa the tube-mill enthusiasm has spread to other continents, among the latest adherents being the Sao Bento in Brazil, as well as various concerns in the United States, Mexico and the Central American republics. The zeal, however, which led to the proposed use of these machines for crushing

coarse particles is now being tempered by experience; and, as I prophesied in the *Engineering and Mining Journal* a year since, it is now being seriously considered whether some form of pan or mill is not the more suitable machine for the breaking down of the coarser quartz particles. Denny, indeed, proposes to effect this preliminary work in a machine of the Chilean mill type and even to avoid the use of stamps entirely.

Regrinding.—To put it briefly, the tube-mill is now settling down into the position of a profit-increasing appliance; but at any moment the commencement of operations on the large scale laid down in Johannesburg may render output increase once more the most prominent feature. At the moment we hear less of African tube-mill work, because local energies are concentrated on the putting into commission as rapidly as possible 30 or 40 more of these machines of the largest size.

Of results achieved up to date, we have the Glen Deep figures that three mills per 100 stamps will effect a duplication of output (the original estimate of five mills was evidently a safe figure), and give an extra extraction of 1s. 6d. to 1s. 10d. per ton (see chairman's speech at the last annual meeting of the Rand Mines, Limited), with an estimated increase of output, without extra cost, of £231,176 per annum for the thirteen companies referred to.

In this connection it is interesting to know that one 5-ft. by 22-ft. tube-mill at the Glen Deep has actually proved itself capable of increasing the output of 20 stamps to 200 tons per day, using an 8-mesh screen in the battery, with a finished product from the tube-mills of under 2 per cent. retained on a 60-mesh screen. Further work showed that the same tube-mill dealt with the output of 40 stamps through a 10-mesh screen for a total output of 336 tons per day.

When using the 64-mesh screen, two-thirds of the total tonnage crushed was reground in the tube-mill; and three tons of material was passed or repassed through the mill for every one ton of output of finished pulp, or one ton intake of coarse sand.

The Robinson Deep enjoys the distinction of being the first mine at work on the Rand showing from practical working the amount of increased profit on output obtainable with more than one tube-mill at work. Last year, as stated in the *Engineering and Mining Journal,* Mr. Caldecott anticipated an increased extraction of 5 per cent., showing an increased clear profit of 1s. ½d. per ton. It appears from the speech of the chairman of the Consolidated Goldfields that with two tube-mills only the output has been increased by 6½ per cent. and the monthly profit by over 19 per cent., or over £5,000 a month, which is over 3s. per ton treated. From this it appears that Mr. Caldecott's estimate was entirely on the safe side.

This is a very fine showing, and if the other African mines can thus increase their profits by anything like £60,000 a year there must be a bright future in store for the industry. In this connection it appears that another

of the Goldfields group, the Knights Deep, has increased its output by no less than 35 per cent., using three tube-mills per 100 stamps, with a reduction in costs over milling the same quantity by stamps only (using 50 extra stamps) of 1s. 10d. a ton. Clearly the point to aim at is not merely the lessened cost of milling (arising from crushing by tube-mills plus stamps), but the increased extraction resulting from the finer comminution of the ore particles.

Leaving Africa for the moment and glancing elsewhere, it appears that the Waihi company, in New Zealand, is able, with two tube-mills, to replace the 40-mesh screens on its 90-stamp mill by 20-mesh screens, with a resulting output increase of fully 30 per cent. It is too soon, however, to gage the full effect of this work on the future of this famous mine; until the most economic conditions of mesh and extraction are determined, the work done must be regarded as more or less experimental; but meanwhile the Waihi company has ordered more tube-mills.

At El Oro, in Mexico, an equally attractive result has been secured. The chairman refers to a better extraction of 13 per cent., or an additional net profit of £65,000 per annum.

From Australia on the other hand, curiously enough, we have a controversy as to whether the pan is not a more efficient instrument than the tube-mill for sliming. The possibility of such a controversy was pointed out in the *Engineering and Mining Journal* last year, and it is a most remarkable feature of Kalgurli work that their tube-mill practice (Hannan's Star excepted) appears to be decadent, rather than progressive. As a result of an idea, they cut down all their mills to 11 ft. or 13 ft., and it so happens that the only mill doing really good work (the mill that was most quoted in the discussion before the Institution of Mining and Metallurgy) is the old original Hannan's Star mill, with its big discharge and long cylinder. To show the fallacy of the recent Ivanhoe tests, it is only necessary to compare the work there done with that elsewhere, and to remember that anyone, however incompetent, can make a mill run badly, but that such work is no criterion of the effective capacity of the apparatus. Apart, however, from the bad work of the Ivanhoe tube-mill, if they had only taken the normal costs of flints and liners, instead of over double the amount, the figures given would have been as much in favor of the tube-mills as they now appear the other way; it must be remembered that this normal cost, of 5d. a ton, is that shown by the Hannan's Star for crushing an incomparably harder grit. The Ivanhoe ore is a soft ore. In spite of this difference, however, the Hannan's Star mill (not a thousand yards away) is actually making a 25 per cent. better output per horse-power. Pans may or may not have a future in the economics of gold-ore treatment, especially on roasted ores; but such inefficient tests as those under notice tend to mislead rather than to inform.

Amalgamation.—One of the results of the impetus given to fine grinding is the discovery that much greater returns can be obtained on the plates from the slime products. The usual practice is to flow the pulp over ordinary silvered plates, and then, after regrinding, over shaking plates. But, even without shaking plates, fine recoveries have been shown. Thus Holloway, in Korea, has increased his amalgamation extraction on an ore containing much pyrrhotite by no less than 50 per cent. Denny shows that, while the plus 60-mesh product yields scarcely any result by amalgamation, the plus 100-mesh yields about 57 per cent. on the tables; the plus 150-mesh no less than 78.3 per cent. and the plus 200-mesh an extraction of 83 per cent. on the plates only. Caldecott points out a similar experience, namely, that the bulk of the amalgamation extraction on the Robinson Deep is derived from the fine material; indeed, he calculates that both the Robinson Deep and Simmer & Jack recoveries on plates amount to almost exactly 90 per cent. of the gold content of the sand passing 90-mesh, assuming the gold in the coarser sand to be untouched.

Filter-Pressing.—Another result of the tube-mill boom is that filter-pressing is more in evidence than ever before. Denny has laid down two large installations of the latest type of Dehne press with hydraulic closing; he reports great success. Filter-presses in Africa have shown themselves capable of treating 2-dwt. slime at a profit, and of returning to the bank over 90 per cent. of the gold content; but the very success of the filter-press makes one all the more eager for a continuous or self-emptying press. Last year no less than five types of such presses were mentioned in the *Engineering and Mining Journal*, but not one of them appears to have successfully survived the twelve months; even the Moore process (of which one has heard so much) appears to be frequently handicapped by some mechanical defect of the plant; but now we have such names as Denny, of Johannesburg; Merrill, Homestake; Butters, Salvador and Virginia City; Banks, Waihi; Hunt, Costa Rica; and Sulman & Ogle, London—all bent on solving this proposition. It is to be hoped that some tangible successful improvement will result. The majority of these efforts are directed to an improvement of the Moore type of basket-frame press; in more than one case, experiment is directed to the hydraulic removing of the press cake from the cloth.

It should be noted here that the growing practice of filling presses by pumps has not infrequently been found to be attended with a lessened extraction, unless provision is made for the introduction of an air jet under pressure into the agitation vats. Probably the air introduced into the Montejus to keep the pulp in suspension was effective in aiding the solution of refractory gold particles; and this additional extraction was missed until an occasional current of air was similarly introduced into the agitators.

Slimes Treatment.—Other than by filter-pressing, this continues mainly on decantation lines. Denny appears to be able to erect at the Van Ryn a continuous decanting plant at a much cheaper rate than the Williams type of plant. The latter is, of course, more complete, and should be capable of higher extraction; but Denny has brought up his returns to 70 per cent. by collecting his residues in a dam and returning the liquor collected there to the plant and the zinc boxes. Gluyas, at the Jubilee, has a continuous agitation by natural flow and settling process; but here, too, probably the difficulty is to get the residues sufficiently free from gold solution, or in other words to recover a large enough proportion of the gold after it is dissolved. Gilmore, at the Santa Francisca, is treating his slime by adding 9 lb. of quicklime to the ton of dry slime (containing 26 per cent. of alumina and resembling China clay); by this means he is able to leach a 4-ft. layer; whereas, without the quicklime, 1 ft. was almost impossible.

West Australian Practice.—This is being very ably expounded just now by Robert Allen (in *Monthly Journal* of the West Australian Chamber of Mines). But, generally, practice has now become so well settled that there is little to record, either in new methods or in lessened working expenses. Costs are now down to under 16s. a ton (2000 lb.) for the Great Fingall (8s. per ton for treatment); 15s. for the Cosmopolitan (2,240 lb.) (5s. per ton for treatment); 18s. 5d. for the Ivanhoe and 20s. 4d. for the Lake View; which mark improvements over those given last year. Detailed costs at the Ivanhoe are as follows: Rock-breaking, 6d.; ore transport, 1d.; milling, 2s. per ton milled; fine-grinding sand, 10d. per ton ground; cyaniding by percolation, 1s. 8d. per ton cyanided; cyaniding by agitation, 4s. 10d. per ton cyanided; filter-pressing, 1s. 3d. per ton filter-pressed. The Ivanhoe total treatment costs are 8s. 9d. per ton milled.

A curious feature of West Australian treatment is the addition of salt to the top of the vats containing concentrate. Mr. Allen states that an extraction of 97 per cent. is thus obtained for a consumption of cyanide of less than 2 lb. per ton. Without the salt, the extraction is from 80 to 90 per cent. for a cyanide consumption of 3½ lb. of cyanide per ton. It is assumed that Mr. Allen means ordinary chloride of sodium and not such a salt as a lead salt.

The percolation of sand does not appear to be meeting with much success; I am informed that even Mr. Moss has now discarded this part of his process, and is adopting complete sliming.

The Merton and Edwards furnaces still hold the field at Kalgurli. A local tip (which is responsible in no small degree for the high extraction obtained from the roasted product) is the addition of lead acetate (say, 2 lb. to each agitator charge of 60 tons) so that any soluble sulphide

formed from an incompletely roasted product may be at once rendered innocuous. At the South Kalgurli, this tip seems to have made all the difference between a $2\frac{1}{2}$-dwt. residue and one carrying 20 grains only.

Undoubtedly the successful feature of the year at Kalgurli has been the good showing made by the South Kalgurli plant since it was taken in hand and equipped by Bewick, Moreing & Co. This plant is now treating $10\frac{3}{4}$-dwt. ore for an extraction of 95.6 per cent. at a treatment cost of 11s. 7d. per ton. Such a regularly obtained extraction is a record which it will be difficult to better. The process used is dry-crushing in ball-mills; roasting in eight Merton furnaces; followed by nine mixing, amalgamating and grinding pans; then agitation and filter-pressing.

As evidencing how vigorously dry-crushing and roasting persists at Kalgurli, and even advances, it may be noted that all the recent plants have been dry-crushing and roasting; and that the firm which has been so prominently connected with the introduction of wet-crushing and bromo-cyanide is not only responsible for the dry-crushing plant at the South Kalgurli, but is now also proposing to erect a similar installation at the Lancefield, which has already been equipped with a wet-crushing mill for the treatment of the surface ores.

Referring once more to the Ivanhoe tube-mill practice, one is tempted to ask: Why can an African tube-mill (taking a material at least as hard and much coarser) crush, to nearly as fine a mesh, four times as much sand per horse-power as the Ivanhoe does? Is it that the design and proportions of the Glen Deep mill are better? Is the Glen Deep mill better worked? Or have silex linings anything to do with it?

African Practice.—This has already been partly discussed under the heading of regrinding. Denny is now "total-sliming" all his material, and will probably have some interesting figures to furnish ere long. Already he is able to show 94 per cent. extraction, and filter-press residues of 0.16 dwt. only, as against sand residues of 0.7 dwt. He is circulating 0.025 per cent. solutions through his plant, and anticipates 98 per cent. extraction as soon as he gets the full benefit of his recently started tube-mill. There are now no less than 59 of these machines (at work, being erected, or on order) for the Witwatersrand alone.

Helman and Crosse have been experimenting with concentration by Wilfleys. But Caldecott has already shown that a large proportion of the gold occurs in the non-pyritic portion of the tailing and slime, and this must tell against any method of treatment based on concentration; or, in other words, it does not seem desirable to separate the sulphides from the sand in the spitz output, when both products (and not the former only) should be reground.

Stark has met with considerable success with his treatment of cyanide residues (described in the *Engineering and Mining Journal*, last year).

Several mines propose to adopt this method. Mr. Stark is resigning from the management of the Crown Reef and applying himself to the exploitation of his process. The Blaisdell excavator was erected at the Robinson mine, and was run for a short time, but no figures of results have been furnished.

Mexican Practice.—Argall, Butters, Hunt, Gordon Wilson and Oxnam have shown that the era of "chloridizing roasting" is going out, and that fine-grinding has come to stay. Indeed, so great an impetus has the success of this sliming treatment given to Mexican ventures that a number of syndicates are now employed in acquiring old tailing heaps for re-treatment by modern methods. For the treatment of the slimed material, decantation is mostly employed, filter-pressing being held—with notable and successful exceptions—to be too expensive.

Zinc-Box Work and Cleaning Up.—One has heard but little this year of the Taverner lead-smelting method; but Miller, of Sao Bento (who has been using for some years a somewhat similar method devised by himself) gives an interesting account of his successful smelting in crucibles instead of in a reverberatory furnace.

Dr. Kirke Rose describes a simple and ingenious method of refining base bullion by the introduction of oxygen or air. The process needs no special tools, and is easily carried out with a minimum of loss, provided one keeps to the acid slag laid down by the author. Dr. Rose presented a most valuable paper on this subject (to the Institution of Mining and Metallurgy), which should be perused by every cyanider the world over. Edmands describes his bullion refining (from 782.7 to 952.9 fine) by melting at a low temperature and adding 15 per cent. of sulphur. The fluid matte is poured off as soon as the gold has set, and contains 50 per cent of the gold, and 73 per cent of the silver. These values are recovered by fusing the matte with iron at a fairly high temperature.

Pawle, of Borneo, saves gold when cleaning up, by filling the last compartment of his box with coke. This last compartment is cleaned up (every three or six months) by merely rummaging the coke between the hands or revolving in a cylinder, the particles thus produced being smelted. He assays his solutions by throwing in a handful of zinc, and adding hydrochloric acid and lead acetate. The lead precipitate is rubbed into a pill in the palm of the hand and cupelled. When testing by Moir's method, he uses Lovibond's calorimeter.

There has been but little of interest in new processes. Professor Ramsay's report on the extraction of gold from sea-water created some interest—another example of the formation of conclusions on insufficient premises.[1] The Elmores are understood to have another concentration process (based on the floating off of the metallic particles by means of gas bubbles generated

[1] Don, *Trans.* American Institute Mining Engineers, XXVII, p. 615.

by an acid solution), and to seek to improve on the Broken Hill processes by a characteristically ingenious modification; the operation takes place in a vacuum, which thus increases the effect of the buoyant gas. One also hears of a process for treating auriferous antimonial ores, which is yielding 90 per cent. extraction to cyanidation after a suitable roast and wash.

Gold Dredging in 1905.

By J. P. Hutchins.

During the past year this flourishing business has had considerably more than a proportionate share of success in the world of metallurgical operation; and it is now in a healthy and progressive condition.

A growing realization of its unique advantages, and a general conception of its unusual possibilities, have contributed largely to this result. Dredging for gold is less in the sphere of hazardous chance, and more within the realm of commercial certainty, than is any other form of precious-metal mining. Placer mining in general, and particularly those operations which include the exploitation of shallow alluvion, have the advantage of being almost, if not quite, on the plane of commercial enterprise. The risk of failure is slight, if the preliminary exploration is directed by skilled engineers and the subsequent exploitation is directed by careful managers; but an unusually high grade of engineering skill, experience and judgment are necessary. The seeming simplicity of the examination of alluvion and of the operation of dredging machinery has often resulted in the employment of amateurs, and usually with disastrous result. Comparatively shallow placers are essential for success in dredging. The ease, cheapness, quickness and great reliability of the prospecting methods applicable to such deposits have called the attention of capital to dredging. The determination of ore-reserve (so vitally important in all mining operations) may be accomplished in its entirety, beforehand and exactly. Such knowledge, when combined with a true appreciation of the characteristics of alluvion, lead to results which can be predicted with certainty.

Results of Prosperity.—The world's production of gold by the dredging method during 1905 is probably less than 5 per cent. and possibly less than 3 per cent. of the total. The reasons for its seeming undue prominence are several. The most potent is the comparative lack of hazard; the success of dredging is its own recommendation. As a result, conditions are assuming a phase which, though not necessarily bad, is at least conducive to that bad condition which makes "booms" possible. Australia is now recovering from the reaction following such a boom in dredging years ago. The volatile inhabitants of Argentine Republic have recently suffered from an excitement imported and industriously stimulated by New Zealanders. Harm has been done although they had sincere intentions; and there is a danger of similar disaster as in Australia, where enthusiasm, insufficient

prospecting, with faulty design and weak construction, caused many failures. The possibility of more booms is not remote; the investing public cannot be too carefully warned against them. In the United States there has been comparatively little illegitimate promotion. Prospecting methods do not readily lend themselves to the unscrupulous. This is due to a uniform and standard procedure, with generally accepted conceptions of prospecting, in which the employment of skilled engineers has assisted. The honesty and integrity of those engaged in all phases of dredging has had its effect on the general prosperity of the business. As a general statement it can be said that, where dredging has been unsuccessful during the past year, there has been a woeful lack of careful prospecting and good management.

Failures and the Causes.—There have been failures, but in every case the causes are easy to determine. Thus unalloyed prosperity has not attended some companies in Australia. Elsewhere the use of "standard" dredges, in areas requiring unusual design and construction, has resulted badly. These dredges are "standard" only because they have been evolved with great success in peculiarly favorable and easy environments. Until it is more clearly understood that, for instance, the Oroville dredge (probably the most successful type) is not a mechanism universally adapted to the successful working of *all* auriferous alluvions, disaster is bound to occur. Examinations by amateurs, and purchases of "stock" dredges from manufacturers who are frequently unwilling to modify the design and construction to suit particular conditions, are responsible for other failures. Nepotism in dredging companies as in insurance corporations has been an expensive luxury.

Modification of Prospecting.—During the past year attention has been directed to the necessity for conducting examinations so as to rectify the incongruity between prospecting and dredging. Shaft sinking, where not impossible nor of prohibitive cost, has been more generally preferred in determining value and in investigating other characteristics of alluvion. The unquestioned advantage of this method (over that of making test-pits of small diameter with drilling machines) has been more generally recognized; it can be predicted that in the future there will not be such a wide difference between prospecting and dredging values. The added advantages of easy and thorough inspection of the various characteristics of the gravel section and bottom (so important in the investigation) are such as to justify the extra expense. In the examination of subfluvial, sublacustrine and submarine alluvion, drilling machines are absolutely necessary; in the preliminary investigation of areas unsuited to shaft sinking they are useful. However, they have not occupied their former exclusive position as being indispensable to all prospecting of dredging ground.

It is seldom that prospecting is done too thoroughly; but in several in-

stances, during the past year, examinations have been so conducted and at such cost that they can only be called extravagant affectations of accuracy. In this, as in other procedure, splitting hairs is of no use. In the examination of lode mines there is frequently ample justification for large expenditure in sinking and driving. The expenditure in prospecting dredging ground is lower than is permissible in lode mining. Overstepping this limit is bad practice. Thus the cost for a superfluous number of test-pits can much better be applied to the installation of dredging machinery and to a working test. In some cases unnecessary disbursements have been made of a magnitude that would install a complete dredge of large capacity and high efficiency. Such practice is pernicious affectation.

Prospecting cost per foot of depth has not varied materially during 1905. Shaft sinking, in a majority of cases, has proved more costly than drilling. It has varied from $1 to $8 per foot, the smaller figure in shallow ground requiring no timbering or pumping. According to recent figures, prospecting in California by drilling machine varies from $1 to $2.50 per foot[1].

Expansion of Dredging.—During the past year the number of gold dredges operating in the world has increased to nearly 500. Investigation brought new cases within the horizon of dredging possibilities; operations are being conducted from Tierra del Fuego (where large volumes of 16c. dirt have been found), to the Arctic circle (where a tenor of more than $1 per cu. yd. has been saved). Large districts in the temperate and tropical zones of South America (Bolivia, Brazil, the Guianas, and Colombia) have been included. They seem to possess the basis for future operations of considerable magnitude. In Alaska and Klondike, the experience of 1905 indicates a more hopeful future. Until recently, because of unfavorable conditions, due particularly to a hostile climate, these districts were supposed to be unsuitable for dredging. Similarly, several years ago frozen gravel was thought to be unworkable by the hydraulic method. In dredging ground, where any considerable extent of frozen material is encountered the high cost requires that there be high value in the ground. Investigation and exploitation have proved the existence of large areas which are but little or not at all frozen. Thus, in one case in the Klondike valley, a dredge with 7-ft. continuous buckets, working alluvion 25 to 30 ft. deep, encountered no frost. This augurs well; the installation of other modern and well designed plants has a similar significance. Frozen condition of alluvion added about 40c. per cu. yd. to the working cost, for thawing.

Provincialism in Dredging Practice and Dredge Design.—This unfortunate condition has been more noticeable than is warranted in these days of progress; thus design of close-connected buckets has delayed progress. The devotees of the headline in their blind adoration refuse to see the advantage of the spud.

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1 "Gold Dredging in California," *Bulletin* No. 36, California State Mining Bureau.

Several years ago when "paddock," or inland, dredging (with its accompanying difficulties of disposal of fine) was in its inception in the Oroville field, I visited Bannack, Montana. The particular problem there had been the evolving of an efficient dredging pump; elsewhere they called it miraculous and impossible, and went on trying to solve the problem in their own ways. They ignored, to their disadvantage, the long experience at Bannack. This spirit is manifest in numerous other ways; it may be fostered by manufacturers who are often unwilling to modify design and construction to suit particular conditions; this may be aggravated when dredge operators become partisan.

Variation in Practice and Design.—During 1905 practice has apparently continued to vary more widely than ever. In the design of gold-saving areas, and with this object in view, we have made strenuous efforts and we are justified by seemingly good results. In this respect of gold-saving arrangements, the Australians and New Zealanders have gone through the list of gold-saving devices; recent practice contemplates either a table-area of about one-sixth that of our dredge, or no tables at all, using the sluice-box alone. It is difficult to reconcile such contrary opinions. Gold, both here and in the antipodes, has the same aspect when seen from the view-points of saving, specific gravity, fineness and amalgamability (the potent factors in extraction); hence gold-saving methods are practically coincident here and in Australia. Does it seem possible, then, that such different conclusions can be the result of correct deductions from careful practice? The dredge with large table areas (either on the main hull in co-ordination with screens having perforations of about 0.5 in. in diameter, or on an auxiliary scow in conjunction with a grizzly as the hydraulic mining undercurrent is used) is considered the perfect gold-saver on the Western Continent.

The experience of the United States with the device which is now preferred in New Zealand and Australia (the sluice-box alone) has created doubt as to its efficiency. To cite one instance, material which had been originally passed through 120 ft. of sluice, on redigging and passing through 30 ft. of sluice of the same type, yielded about as much gold as at first. This is not entirely determinative, however, as the possible presence of material which disintegrates with time and exposure would vitiate any positive conclusion. Australians and New Zealanders (by the side of whom American dredgemen are as infants) seem to have quite different characteristics, in that they have tried more varied gold-saving devices than we have, and through a much longer time. Amalgamation as an aid in saving gold is in disfavor among them. At first, some are tempted to conclude, as experience is the best teacher, that their own practice is best. However, when one reads that only one-half as much gold is recovered by the floating dredges as is saved in other placer operations, in areas of the same characteristics in Australia, all is in doubt again. Here again confusing factors

make conclusion hang. It can be said, as a general statement, that even though conditions in Australia, New Zealand and America are somewhat dissimilar, yet such contradictory conclusions can be the result only of haphazard and crude manipulation.

Sampling of Tailing.—The chaotic conflict of practice (more obvious than ever in 1905) is almost entirely due to the ignorance of the approximate extraction accomplished by the gold-saving apparatus. Prospect values and gold extraction per cubic yard have been considered in detail. The cost of handling material per cubic yard has been minutely tabulated in cost sheets with numerous items; extraction, repairs, lost time, etc., have been examined microscopically. Deductions as to extraction have been made simply by comparing results of clean-ups, with results of prospecting; this procedure has shown that the relation of extraction to prospecting varies from 40 to 250 per cent. and *vice versa.* This is analogous to ascertaining the extraction in vein mining, by comparing clean-up with assay value, without any sampling of tailings. Such bad practice is unquestionably responsible for the present condition; until it is known what proportion of the gold tenor is lost in tailing, numerous different gold-saving devices will be used (locally, for no other reason than custom); gold-saving practice will vary with the personal factor.

Modifications of Design and Construction.—No striking novelties of design or construction have been introduced during 1905. Excavating, screening and tailing apparatus have approached similarity in design and construction in all parts of the world; however, this convergence is taking place slowly. In sluicing and gold-saving, the tendency has been to diverge. Amateurs conducting placer mining with crude incompleteness have always been responsible for seemingly contradictory results, which are likely to stagger an inquirer for exact information. Dredging is handicapped by this unfortunate condition. In a period when all metallurgical operations have been striving for refined methods, such uncouthness is unpardonable.

To be sure, there has been an improvement, and dredges have excavated larger volume and at less cost than ever before. High saving has been accomplished principally because of the new design of dredges, which includes the better co-ordination of the various phases. Early designs were notoriously deficient in this respect; and a constant overloading of gold-saving devices, due to insufficient capacity, resulted in loss. Dredges are now being constructed to dig 60 ft. below the surface; this means that it is possible to handle alluvion more than 75 ft. deep (by having a bank more than 15 ft. above water level).

The centrifugal tailing-elevator has proved its worth in New Zealand and Australia during the past year[1]. Its adoption in America is yet to be chronicled. The first steam-turbine for gold dredging was installed during

[1] "A New Centrifugal Elevator," by W. Peck, *Engineering and Mining Journal*, Aug. 5 (1905), p. 199.

1905; its use will be of material assistance in many instances. It was adopted in the Klondike, to economize the expensive fuel and to allow larger deck room on the dredge. A short description of this new plant is of interest. The dredge has 7-ft. close-connected buckets, with a record of 3000 cu. yd. per day. Its rated motor capacity is 290 h.p., of which less than one-half is normally used. The power plant includes boilers of combined capacity of 450 h. p.; a 600-h. p. steam turbine (this size was installed as it was intended to operate several dredges) and surface condensers. Electricity at 2300 volts was generated and transmitted to the dredge, where transformers reduced it to 440 volts. The entire installation was completed and in operation in 43 days; it ran satisfactorily, even at a temperature of 2 deg. below zero, until continued cold weather caused a shut-down. This is unusually good work, and is highly commendable.

A marked progress is manifest in the disposition of operators (with wide experience) to prefer trommels to shaking screens. The superiority of the revolving screen is shown in economy of power and maintenance cost, due to its mechanical features, and to its better work as a screening device; this latter advantage is due to its disintegrating effect.

The general tendency during 1905 was to build dredges larger, with greater proportionate strength of parts and increased capacity. That larger buckets can excavate to better advantage in alluvion containing boulders, clay or cement has been more generally acknowledged. Various bad mechanical arrangements (like that by which stacker belts were driven from the lower end) have been eliminated in the newer designs. A general improvement is noticeable.

Some of the recent years were periods of retrogression by reason of the general bad design and bad construction of dredging machinery. The ability of the dredging industry to prosper despite such handicaps indicates its flourishing condition. There have been several installations of freak types, but they have accomplished no general good.

Modifications in Manipulation.—There has been a refinement of method particularly in the prevention of wasting adjacent virgin ground. More care in gold-saving has been noticeable, and considerable progress has been made. A great divergence of manipulation still occurs in methods of *holding* dredges; however, the respective merits of spuds and head-lines have been better demonstrated than ever before.

Adverse Legislation.—The dredging industry has assumed such a magnitude as to threaten, in a slight degree, contiguous and adjacent interests; but no hostile associations and legislation (such as that which resulted from hydraulic mining in California) have developed. In Australia, however, dredging operations have been regulated (but not materially obstructed) by legislative enactment. In California, ill-advised agitation by hysterical devotees of æstheticism, and action by sycophantic black-

mailers, have caused some annoyance; but no harm has resulted or can result to any industry doing such inestimable good with such slight accompanying damage.

Merger of Dredging Corporations.—The first important instance of what undoubtedly will be a series of mergers has been consummated in 1905. More than most metallurgical operations, dredging invites consolidation; the results of this have been good. The time when each dredge was an independent individual in a dredging community, with its own manager, city officer, superintendent, foreman, machine- and blacksmith-shop, is passing. Cost is more likely to be materially reduced by pooling than by improvement in design, construction or method.

During the past year there has been a slight reduction of costs due particularly to the incorporation of several companies under one management, and in a lesser degree to the building of larger dredges with increased capacities. Cost varies greatly—from 3c. per cu. yd. to more than 50c. per cu. yd. in Klondike and Alaska. Cheap work can be done only in temperate zones. The obstacles, incidental to tropical or frigid climates, and which only hinder other mining operations, are serious when they concern this industry. Dredging cost is very sensitive to environment.

Cyanidation During 1905.

By Charles H. Fulton.

The year 1905 marks distinct progress in cyanidation in almost every one of its many phases and applications. The chief improvements, however, are to be found in the treatment of slime by filter-pressing; in the greatly extended application of the process to ores whose value consists chiefly in silver; and in the successful treatment of cupriferous gold and silver ores.

Silver Ores.—The chief development of this branch of cyanidation has taken place in Mexico, although some plants are in operation in the United States, the most prominent one being the Butters plant on the Comstock Lode in Nevada. The process is employed in Mexico on such ores as were formerly treated by pan-amalgamation and by the patio processes, and also on the lower grades of this same general type of ores which formerly could not be treated economically. The silver in these ores exists chiefly in the form of argentite, stephanite, a little tetrahedrite, and some native silver, with small amounts of chloride and bromide. In general, these ores are more complex than most gold ores, in that they frequently contain minerals yielding appreciable quantities of antimony, arsenic, bismuth, etc., which make their treatment more difficult.

T. H. Oxnam[1] describes the cyanidation of silver ores at the Palmarejo mine, Chihuahua, Mexico. The ores are highly silicious, and contain silver

[1] *Engineering and Mining Journal*, Aug. 19, 1905, p. 297; Aug. 26, 1905, p. 339.

in the form of argentite, accompanied by some stephanite and a small amount of native silver. Pyrite, manganese oxide, and calcite occur, as well as small quantities of antimony, arsenic and bismuth. The ore is crushed by stamps, and the pulp is concentrated on Wilfley tables, after which it is classified into sand and slime in a somewhat crude way, and then cyanided. The sand contains 16 oz. silver and $2.85 gold, Mexican. The slime contains 19.25 oz. silver, and $4.35 gold, Mexican. The extraction from the sand is 96 per cent. of the gold and 54 per cent. of the silver. The extraction from the slime (which is treated by agitation and decantation) is 51 per cent. on the silver and somewhat less for the gold than from the sand. The strength of the solutions used on the sand is 0.75 and 0.25 per cent., respectively, sodium cyanide. Somewhat weaker solutions are used on the slime. The sand is leached for 10 days; the consumption of cyanide is 2.98 ℔. per ton treated. The consumption of zinc is 0.96 ℔. per ton. The consumption in the treatment of slime is 3.56 ℔. NaCN, equivalent to 4.42 KCN. The cost of cyaniding sand is $1.52; of slime, $2.66 U. S. per ton.

G. N. Miller[1] describes briefly the operation of a cyanide mill in western Chihuahua. The ores contain two-thirds of their value in silver, and one-third in gold. After crushing by stamps and concentrating on Wilfley tables, the pulp is classified into sand and slime; the sand is treated by percolation; the slime by filter-pressing. A total extraction of 84 per cent. is obtained from the material cyanided, about equally good extraction being obtained for the gold and silver.

Alfred Chiddey[2] describes the cyanidation of a silver ore at El Salvador, Mexico. The ore is crushed by arrastras, the pulp passed over amalgamated plates, and then separated into sand and slime. The sand contains 13 to 15 oz. of silver and $3 to $5 in gold. The strong and weak solutions have strengths of 0.4 and 0.2 per cent. NaCN, respectively; 85 to 90 per cent. of the silver is extracted, and 90 to 92 per cent. of the gold. The consumption is about 2 ℔. of NaCN per ton of material treated.

J. W. Malcolmson[3] describes briefly the cyanidation of silver-gold ores in the Guanajuato district, Mexico, more particularly at the Sirena mill. The pulp, after concentration on Wilfley and other tables, is separated into sand and slime. A total extraction of 87 per cent. is made by concentration and cyanidation, 75 per cent. being extracted by cyanidation. Large volumes of solutions are employed in the treatment. In the Guanajuato district the gold varies from 15 to 70 per cent. of the combined gold and silver values.

H. G. Elwes[4] discusses the application of cyanidation to the lower-grade

[1] *Engineering and Mining Journal*, Aug. 26, 1905, p. 344.
[2] *Ibid.*, June 1, 1905, p. 1053.
[3] *Ibid.*, Sept. 23, 1905, p. 529.
[4] *Ibid.*, Mar. 16, 1905, p. 515; July 22, 1905, p. 109.

silver ores of Zacatecas and other districts of Mexico; and the treatment of rich argentiferous concentrates by cyanidation after chloridizing roasting at a low heat (dull red) and for a short period of time (15 to 30 minutes).

The main points, in the cyanidation of silver ores, on which most metallurgists of experience agree, are:

(1) That a comparatively long time of treatment is essential, ranging usually from 10 to 25 days. This seems reasonable owing to the fact that silver compounds are more difficultly soluble in cyanide than gold.

(2) That a thorough oxygenation is necessary. This is obtained in most instances by a double treatment; by the withdrawal of the solutions from the vats, and by the re-aëration of solutions, etc. Granting that the reaction involved in the solution of silver by alkaline cyanides is similar to that for gold, and that the oxygen dissolved in the solution is that which is available, it would seem clear that (owing to the much larger bulk of metal to be dissolved, and the more complex nature of the ore as regards base metals present) this oxygenation is essential. In this connection, it is possible that such reagents as barium dioxide, sodium peroxide, etc., and bromo-cyanogen will find a much larger field than was ever probable for them in the cyanidation of gold ores.

(3) That somewhat stronger solutions than are used in the cyanidation of gold ores must be employed. The solutions, weak and strong, range from 5 to 16 ℔. NaCN, per ton of solution (which, to be expressed in terms of ordinary cyanide, must be increased by at least 25 per cent). Generally speaking, in the cyanidation of gold ores, 5 to 6 ℔. of KCN is considered a comparatively strong solution, and solutions of 1.5 to 2 ℔. per ton are quite ordinary.

(4) The consumption of cyanide (as might be expected in the cyanidation of silver ores) is much higher than in the treatment of gold ores. It varies, in the cases recorded, from somewhat under 2 ℔. to 3.56 ℔. of NaCN, which must be increased by 25 per cent. to express it in KCN. While no general figures for consumption of cyanide for gold ores are available, from 0.4 to 0.8 ℔. KCN per ton of ore is frequent, and 1.5 ℔. is considered high.

(5) The precipitation of silver by zinc-thread or dust presents no difficulty and is practically complete. As the bulk of metal to be precipitated in the treatment of silver ores is many times more than that to be precipitated in the treatment of gold ores, it might be expected that the consumption of zinc would be increased. This, however, is not so. The zinc consumption per ton of sand treated (mentioned by Mr. Oxnam) is 0.96 ℔. per ton of sand treated or 1.75 oz. per oz. of silver precipitated. At Carbonate, S. D., a small tailing plant treating silver ores a few years ago gave a zinc consumption of 0.86 oz. zinc for every ounce of silver precipitated. The zinc consumption in the treatment of silver ores is practi-

cally the same as that in gold ores (when the figures are expressed per ton of ore treated), ranging from about 0.55 to 1.5 lb. according to the nature of the ore and the amount of solution to be precipitated; but, when stated in figures expressing the consumption per oz. of metal precipitated, gold requires from 9 to 17 times as much zinc as silver. The facts as stated are probably explained by the greater concentration of the silver solutions; or, in other words, that the zinc consumption is practically a constant per ton of solution to be precipitated (irrespective of the metal contents of the solution), within such limits as are common in cyanide practice.

Treatment of Slime.—Without question, the greatest improvement in cyanidation within recent years, and one which holds out great promise, is the invention and perfection, by C. W. Merrill, of a filter-press which can be rapidly and automatically discharged without opening the press. The press is discharged by sluicing. The press in general is of the "flush-plate and distance-frame" pattern, except that the units are much larger than in ordinary slime presses, having an area of 25 sq. ft., more or less. The chambers of the press are filled from a continuous channel, running along at the median line at the top of the frames and communicating with each filter compartment. Solutions are admitted from a channel in either of the upper corners of the frames. At the middle of the bottom of each frame, and passing through the filter-plates, is a continuous channel within which, with ample room to spare, is a supply pipe for water under pressure. The pipe is provided with nozzles projecting into each compartment. The pipe can be revolved through practically 180 deg., so as to play a stream of water upon all portions of the slime-cake in the chamber to be discharged, washing it down into the annular opening formed by the continuous channel through the plates and frames, and the water supply pipe. When the press is being filled and leached, the discharge channel is sealed at both ends. A modified form of the press has the water pipe and nozzles in one of the lower corners; the continuous discharge channel in the other lower corner; and each frame is provided with openings on each side, closed by doors to permit access to each filter-frame. The usual channels for withdrawing filtered solutions are provided. The press is covered by patents (798,200, etc.). It has great capacity and does away with the greater part of the cost of labor of discharging, one of the chief items in the cost of filter-pressing. Presses of this type have been in operation, experimentally, in the cyanide plant of the Homestake Mining Company, at Lead, for nearly a year, on extremely low-grade slime, having a value of from $0.80 to $1.20 per ton. That they have been commercially successful is assured by the fact that the Homestake company is at present constructing (on McGovern hill, near Deadwood), a filter-press plant to contain about 25 presses at a cost of close to $300,000. About 4000 tons of ore are crushed daily by 1000 stamps. After amalgamation this pulp is sep-

arated into approximately 2500 tons of sand (treated at the Lead and Gayville sand plants), and 1500 tons of slime (at present wasted, but soon to be conveyed by iron-pipe lines to the slime plant on McGovern hill).

The Moore process is finding increased application (*Trans.* A. I. M. E., Vol. 35, "Crushing in Cyanide Solution in the Black Hills"), and is being gradually perfected. It has recently been installed by the Standard Consolidated Mining Company at Bodie, Cal., and has been in operation during the year at the Lundburg, Dorr & Wilson plant at Terry, S. D., with success. It is also being installed by the Puritan company near Deadwood, S. D., and by the Ernestine Mining Company, at Mogollon, New Mexico. The Liberty Bell Mining Company (which has recently installed tube-mills) treats its entire product by the Moore process. At Bodie the process treats the tube-mill product, making an extraction of 85 per cent.

The Mumford slime-process has been experimented with at the Robinson Deep mine in the Transvaal. It is an electro-cyanide process; the slime (after agitation with very dilute cyanide solution in a vat) is transferred to a revolving cylinder lined with an amalgamated copper plate which acts as the cathode, while the anodes are carbon rods in the heads of the cylinder. It is difficult to see the advantages of processes of this general type (such as the Pelatan-Clerici, Riecken, and Hendryx), which precipitate from the pulp direct. Conceding that the precipitation is good (which, however, is distinctly not the case), the loss of soluble cyanide, even though decidedly low-tenor solutions are used, will always be urged as an objection to these processes.

The Ogle continuous filter[1] is a light-iron annular frame-work on which are 16 hollow filter-trays, suspended on trunnions. The frame-work revolves horizontally, the trays becoming charged with approximately ½-in. cakes; at a certain point, suction is then applied automatically to the tray; and, after traveling practically a complete revolution, the tray is tipped by an automatic tripper to discharge it, the tray immediately being refilled. Provision can also be made to wash the filter cake with solution and water during part of its travel.

Slime.—There was considerable discussion during the year as to a proper definition of "slime." A. W. Warwick[2] defines slime as "that portion of the crushed ore which, owing to its containing combined water, possesses the property of forming colloid hydrates when mixed with water." What is known as slime in most mills will contain these colloid hydrates, mixed with a considerable percentage of fine gritty quartz or of similar material. This addition is in many cases essential in order to permit the slime being treated at all, by filter-pressing for instance. True slime, in some cases, would make a perfectly impervious mass.

[1]*Engineering and Mining Journal*, Feb. 23, 1905.
[2]*Western Chemist and Metallurgist*, I, 2, p. 51.

Cyanidation of Cupriferous Ores.—Considerable attention is being paid to the cyanidation of gold and silver ores that contain copper up to about 1.5 per cent. as a maximum. Amounts of copper anywhere approximating this percentage prohibit ordinary cyanidation under most circumstances. Jarman and Brereton[1] experimented with the ammonia cyanidation of copper ores, employing KCN and ammonia (practically the Hunt process), and with KCN and ammonium chloride. It is found, for the Hunt process, that the relative amounts of KCN and ammonia must be carefully adjusted in order to get good results. When the proportion is such that a minimum of copper is dissolved, it is found that the maximum of gold goes into solution. H. L. Sulman considers that the solvent for the gold in this case is a cupro-ammonium cyanide, having no cyanogen combined with the alkali metal; and that the solvent action is mainly due to the nascent cyanogen derived from the cupric cyanide of the cupro-ammonium compound.

An interesting (and, so far as I know, the only) example of the cyanidation of a cupriferous gold ore is that of the Bagdad Chase Gold Mining Company at Camp Rochester, in the Mojave desert, Cal. The ore is an altered eruptive rock, silicified, and containing iron oxide, malachite, and chrysocolla, with a gold value of $20 and upward. Slacked lime is added to the ore in the tanks for neutralization, and also 1 lb. of ammonium chloride for every ton of ore. The ore is then leached with a strong solution of KCN, dissolving copper, silver and gold. The solution is precipitated by zinc-thread, the high tenor of the solution in KCN preventing to a large extent the precipitation of copper in the zinc boxes. After leaving the zinc boxes the solution is led to a tank, where the copper is precipitated by zinc dust, agitation for several hours being necessary to complete the precipitation. The amount of zinc dust required is from 1 to 1.5 times the amount of copper present in the solution.

Sodium and Potassium Cyanide.—Whether sodium or potassium cyanide is to be employed is still actively discussed. When the practically pure sodium cyanide was put on the market, its high value in cyanogen, its great purity, and its supposed greater solvent power were expected within a short time to replace the KCN or the mixed salts. In the United States, however, a reaction has come in favor of the KCN, or the mixed cyanides. A number of plants in the Black Hills and in Nevada have replaced the NaCN by the KCN and the mixed salts, on the ground that the NaCN is not quite as efficient as a solvent. Many plants, however, in this country prefer the NaCN; notably the Kendall, Mont., plants. In many Mexican plants, NaCN is in high favor, especially in those treating silver ores, which claim a higher solvent power and an appreciable saving in cost of transpor-

[1] *Trans.* Institution of Mining and Metallurgy, Feb. 16 March 16, 1905.

tation. NaCN deliquesces considerably; and, in moist climates, boxes once opened must be used rapidly.

Crushing.—Finer crushing in many districts is noticeable, and the tube-mill is finding favor in this country. The Liberty Bell mine, at Telluride, has installed tube-mills; the Standard Consolidated Mining Company, in California, has also installed one. The United States Reduction and Refining Company, at Colorado City, is putting in an experimental tube-mill plant for Cripple Creek ore. Huntington mills are installed in two plants to regrind coarse material from stamps. Crushing in cyanide solution is finding increased application in this country wherever the nature of the ore permits it. The Standard Consolidated, at Bodie, uses this method, crushing in weak cyanide solution and amalgamating over plates. The new mill of the Tonopah Mining Company, now being constructed at Goldfield Junction, Nev., will also crush the ore by stamps in cyanide solution. Many other mills are using this method.

General Review by States.—While the number of new mills erected during the year is somewhat less than within the past few years, the amount of ore subjected to cyanidation has increased.

In California, the Standard Consolidated has been in steady operation; in the desert region in Kern county, the Queen Esther, Echo and Karma have treated considerable ore. Two of these mills amalgamate over plates, and cyanide the sand and slime, one of them crushing in cyanide solution. The decantation process is used for slime treatment. Most of the other well-known mills in California have been in operation during the year. In Sierra county, the Sierra Buttes mine has installed a cyanide plant.

In Colorado, the Camp Bird, Smuggler-Union and Liberty Bell mines have been in commission; in Gunnison county, the Taylor Park mine has constructed a 200-ton concentrating and cyanidation plant. It is stated that the Independence company has acquired the Dorcas cyanide plant for the treatment of its lower-grade ores. Several small plants are in operation in Boulder county. There are now in the Cripple Creek district six cyanide plants, having a total actual capacity of about 700 tons per day. All of the mills are not in operation, nor do they operate to their full capacity. In September, about 16,000 tons of low-grade ore were treated. The general practice is to crush coarse through a final screen of from 0.2 to 0.375-in. opening, and leach with solutions ranging from 1.5 to 5 lb. per ton. The time of treatment is approximately three days; the cost, from $1 to $1.50 per ton. The Homestake treats very low-grade material, but its commercial success has not as yet been demonstrated. The range of ore values for the other mills is from $3.50 to $6 per ton.

The milling practice at the Camp Bird, as carried out at present[1], con-

[1] S. L. Goodale, *Engineering and Mining Journal*, May 4, 1905, p. 850.

sists of crushing with stamps; amalgamating over plates; concentrating on vanners; classifying the vanner tailing by spitzkasten; regrinding the coarse sand in Huntington mills; and passing the middling and fine, with the product of the mills, over a second set of vanners. The tailing from the vanners goes to five-compartment spitzkasten, eliminating most of the slime which for the present is stored. The sand is fed to the leaching tanks, by Butters-Mein distributors. Double treatment of the sand is practiced and the time of treatment is practically 20 days; 3 tons plus of solution is applied per ton of sand, the solutions ranging from 2 to 5 lb. of KCN per ton. The extraction of gold in cyanidation is 78 per cent.

In Nevada, a number of new mills have been erected, and are under construction, notably that of the Tonopah Mining Company, at Goldfield Junction. It is a straight cyanidation mill. The ore is to be crushed by a hundred 1050-lb. stamps in cyanide solution; the pulp classified in 10 coarse classifiers; and the coarse product from these to be reground in five 5-ft. Huntington mills to 50 to 60 mesh. All of the pulp is then to go to 20 fine classifiers for double classification; the slime going to nine slime vats provided with agitator arms. The sands go by means of Butters distributors to sand-collecting vats provided with slime overflow-gates, effecting a further elimination of slime. From the sand-collecting vats, the sand is removed by an excavating machine, and is delivered to a belt conveyor, feeding a centrifugal distributor which lays down the sand in a leaching vat. The slime is to be treated by the decantation process, the slime vats being provided with horizontal agitator arms on a vertical shaft. A filter-press is used for clarifying the slime-solution before precipitation. Ordinary zinc precipitation is employed; a reverberatory drying-furnace, with a hearth of 6x10 ft., will be used to dry the precipitate, which is then smelted in tilting furnaces.

In Storey county, on the Comstock lode, the Butters company has been operating its 200-ton plant, on dump material during the year. In Lincoln county, the Janet Mining Company has erected a small plant, using the Holderman filter-tank process. Several small mills are in operation in the Search Light district. The Bamberger de La Mar plant has been in operation throughout the year.

In Idaho, the Lincoln mine, at Pearl, Boise county, has installed a 150-ton plant. The Gold Dust and Bitter Root mines, in Lehmi county, have installed cyanide plants, as has also the Iron Springs Company, in Washington county.

In New Mexico, the Ernestine Mining Company (at Mogollon, in Socorro county) has installed a cyanide plant to treat tailing from a 20-stamp mill, after concentration.

In Arizona, the Congress mine (in Yavapai county), and the Gold Roads

mine (in Mohave county) have been in operation, as well as several other small plants.

In Utah, the Mercur company has operated more successfully than in the year previous; the mills of the State-line district have been in operation.

In Oregon, in Baker county, the Virtue company has installed a cyanide plant; the Mayflower company is operating a new plant.

In Washington, the Nooksack Mining Company has put its 200-ton plant in commission.

In Montana, cyanidation has been very active, although no new plants of importance have been erected during the year. The plants at Kendall have operated largely and successfully during the year. The Kimberly Montana, at Jardine, and all of the other well-known mills in the State have been in operation.

In South Dakota, the two Homestake plants, the Golden Reward, Imperial, Dakota, Wasp No. 2, Spearfish, Maitland, Gilt Edge Maid and Puritan companies have been in operation. The large plant of the Horseshoe company was in operation until destroyed by fire last May. This company will remodel its Pluma plant, and be ready to treat ore in the near future. In the Squaw Creek district, the Victoria and Eleventh Hour companies are constructing dry-crushing plants; the Reliance company, at Annie Creek, will erect a plant. The Hidden Fortune, Alder Creek, and Deadwood Standard plants are not in operation.

Progress in Gold Milling in 1905.

By Robert H. Richards.

An Improvement on Swinging Feeders in Stamp Mills[1].—Paul J. Johnson describes and illustrates with a drawing an improvement on swinging feeders. This improvement is an iron brace bolted on the mortar and supporting the swinging feeder, thus eliminating the bodily swinging of the feeder with each blow of the tappet.

Nissen's Circular Mortar[2].—This is an illustrated description of a circular mortar made by the Nissen Engineering Company, Los Angeles, Cal. The advantages claimed for this mortar are: no round corners, screen always normal to discharge (area of screen 12x39 in.); screen does not receive severe wear. The weight of the mortar is 2400 lb., and of the stamp 1300 lb. The capacity is 8 to 12 tons per stamp per 24 hours through 40 to 20 mesh, with 3 h.p. per unit. The stamp also has special arrangement of cam shaft posts, which are set in a solid concrete foundation, thus eliminating to a large extent the breaking of cam shaft.

Improved Mining Appliance.[3]—The statement is made that Mr. A. M. Robeson at the Village Deep mine, South Africa, has increased his capacity

[1]*Mining Reporter*, Vol. LII (1905), p. 516.
[2]*Engineering and Mining Journal*, Vol. LXXIX (1905), p. 203.
[3]*New Zealand Mines Record*, Vol. VIII (1904-5), p. 409.

per stamp per day from 5 tons to 6.1 tons by the introduction of anvil blocks under the mortar-boxes of the batteries.

Heavier Stamps on the Rand.[1]—The statement is made that the Simmer and Jack mine on the Rand has replaced 1250 lb. stamps by those weighing 1415 lb. The further statement is made that the General Mining and Finance Corporation is making experiments at its mines in which grinding pans are substituted for stamps.

The Finer Crushing of Banket Ore[2].—W. A. Caldecott discusses the finer grinding of banket ores from various tests on the Simmer and Jack and the Robinson Deep ores. He comes to the conclusion that the principal losses occur in the coarser sands with adhering pyrite and in the pyritic portion. He argues that to regrind all sands from battery coarser than 60 mesh would increase the extraction from 90 to 95 per cent., of which extra 5 per cent. 2½ per cent. would be profit.

Fine Grinding.[3]—P. Argall, in discussing a paper by A. W. Warwick in the *Western Chemist and Metallurgist,* makes the statement that, at a recent experiment on the Rand, a 1250-lb. stamp gave, at 0.02 in. screen aperture, a capacity of 5 tons per day; at 0.04 in., 6.5 tons; at 0.05 in. 7 tons, and at 0.08 in. 10 tons per day, and the best all round results were supposed to be made when crushing through 0.08 in. aperture. He further gives his idea of a modern fine crushing plant as follows: Rock breakers to 2 in. cubes; rolls in series, 2 in. to $\frac{1}{20}$ in.; grinding pans $\frac{1}{20}$ in. to $\frac{1}{60}$ in.; tube mills to infinity if required.

Regrinding of Sands.[4]—Charles Butters and E. M. Hamilton give the results of numerous laboratory and mill tests on the regrinding of sands for cyanide treatment. They found that in the case of the sand residues the highest values lay in the coarser material, as shown in the following table:

SCREEN TEST, 2000 GRAMS TAKEN.
Assay 4.06 dwt. gold, 1.72 oz. silver. Total, $5.02 per ton.

Mesh.	Wgt. grams.	% of total Weight.	Assay Value. Gold.	Silver.	Total.
			Dwt.	Oz.	Total
On 30	2	1.90	6.40	2.43	7.78
" 40	36				
" 60	480	24.00	5.60	2.09	6.79
" 80	402	20.10	4.50	1.68	5.46
" 100	206	10.30	3.30	1.47	4.12
" 120	562	28.80	3.12	1.27	3.83
" 150	14				
Through 150	267	13.35	2.10	1.67	2.97
	1969	98.45			

This table suggests that fine grinding is probably the key to the difficulties in sand treatment. The authors further found that in sliming the resi-

[1]*Mining and Scientific Press*, Vol. XCI (1905), p. 421.
[2]*Trans.* Institution of Mining and Metallurgy, Vol. XIV (1904-5), p. 48.
[3]*Mining and Scientific Press*, Vol. XC (1905), p. 357.
[4]*Trans.* Institution of Mining and Metallurgy, Vol. XIV (1904-5), p. 3.

due completely they got a combined total extraction of 95.2 per cent., compared with a combined total extraction of 76.8 per cent. for material ground through 150 mesh. In the large sized tests a Krupp tube-mill was used. The following table may be taken as an example of the grinding of the tube-mill at the rate of 3 tons per hour.

CHARGE No. 7. GENERAL SAMPLE.

Sand before Grinding.		Sand after Grinding.	
On 40 Mesh	10.0%	On 40 mesh	0.0%
" 60 "	26.0	" 60 "	0.0
" 80 "	27.4	" 80 "	1.0
" 100 "	12.4	" 100 "	3.2
" 150 "	11.4	" 150 "	14.8
" 200 "	3.8	" 200 "	11.0
Through 200 "	8.0	Through 200 "	70.0

The authors found that the grade of material as shown by screen tests appears not to vary a great deal with different rates of feed, but that the variation seems sufficient to make a marked difference in the extraction. This may be seen in the following table:

	Charge No. 5.	Charge No. 6.	Charge No. 7.
	1.53 tons per hour.	2 tons per hour.	3 tons per hour.
% passing 200 mesh	78.0 %	72.4 %	70.0 %
% of slimes	38.5	39.5	39.4
Original value	$13.00	$13.47	$13.78
Residue value	$ 1.56	$1.79	$2.04
Value recovered	$11.44	$11.68	$11.74
% extraction	88.0 %	86.7 %	85.2 %

The authors figure out the cost of grinding at the different rates of speed, taking into account power, wear and tear, and depreciation, and they come to the conclusion that the 3 tons per hour rate is the most economical. The figures show that, with the decrease in capacity, the extraction does not increase enough to overcome the increased cost per ton of the finer grinding.

Stamp Mill Capacity.[1]—In this article different methods are suggested to increase the stamp capacity and keep the tailings low in value. The statement is made that the breakers, placed in tandem before the stamp-mill, crushing to about 2 in., increase the capacity economically. Another method is to crush coarse, that is with a screen 8 to 12 mesh, in the stamp battery and follow with a rotary grinder. This last gives increased capacity with a small additional cost.

Tube Mills in Gold Milling.—Some material on this subject will be found under "Progress in Ore Dressing," elsewhere in this volume.

Electrical Driving of Stamp Batteries.[2]—H. J. S. Heather, in discussing a paper on "Electric Driving of Stamp Batteries," presented before the South African Association of Engineers by Mr. Wilms, advocates slow-speed motors (250 r.p.m.) of low number of cycles (25), with a 30 h.p.

[1] *Mining and Scientific Press*, Vol. XC (1905), p. 239.
[2] *Mining Reporter*, Vol. LII (1905), p. 184.

motor for every 10 or 20 stamps, also separate systems for power and lighting. He claims for this system, in comparison with one in which the whole mill is driven by one or two large motors, lower initial cost and less stoppage loss.

Fine Grinding of Gold Ores with Special Reference to the Tube Mill.[1]—There are three types of fine grinding machines: pebble mill, amalgamation pan, and Griffin mill (dry crushing only). The Griffin mill will crush so that 90 per cent. will pass 100 mesh. The following table shows the crushing of the first two, on Kalgoorlie ores, Western Australia:

	Hannan's Star Pebble Mill %	Ivanhoe Pans %
Above 40 mesh	....	....
" 60 "	....	....
" 100 "	0.7	....
" 150 "	4.0	0.5
Below 150 "	95.0	99.5

A description is given of two types of tube-mill, viz., central discharge and peripheral discharge mills. The capacity of the mills depends on the hardness of the material fed. A Kalgoorlie tube-mill 4½x16 ft. handles 38 tons sands per 24 hours. Hard wood linings of a tube-mill last three days (with wet crushing), manganese steel lasts 15 months, silex lining 3 years, and cast iron 3 weeks to 3 months. The pebbles used are Greenland or French flint, and cost $20 per ton. The variables in tube-mill are: rate of feed, thickness of pulp, amount of pebbles and rate of revolution. The cost of grinding at Hannan's Star in a 4½x16 ft. tube-mill is 35c. per ton and at Lake View Consols mine, Kalgoorlie, about 50c. per ton. The grinding is done in tube-mill both by abrasion and impact.

A new definition of the word "slimes" is given, as that portion of crushed ore which possesses the property of forming colloid hydrates when mixed with water. On this basis a new reagent other than lime might be used which would greatly facilitate the treatment of slimes by decantation.

How Expenses Decrease as Stamps Increase.[2]—In an abstract from the *South African Mines* a compilation of working costs per ton of ore milled of some 54 companies in South Africa is given. A summary is given in the following table:

Companies.	Stamps.	Average working costs per ton.
1	Over 200 stamps	$5.37
6	200 stamps	5.41
5	150 to 200 stamps	5.25
10	100 to 150 "	5.47
8	100 stamps	5.80
17	Between 50 and 100 stamps	6.35
7	50 stamps and under	6.11

[1]*Mining Reporter*, Vol. LII (1905), pp. 240, 287, 307, 333. Abstract of a paper read by A. W. Warwick before the Western Association of Technical Chemists and Metallurgists.
[2]*Mining Reporter*, Vol. LII (1905), p. 254.

Although the sequence is not maintained throughout, the figures show that the working costs decrease as the number of stamps increases.

Sizing Tests of Sand and Slime[1].—In discussing grading analyses, H. S. Denny expresses a doubt as to which is the best method of analysis, wet or dry. He bases this doubt upon the reports of four mill managers, two of whom favor the wet and two the dry method. These reports are conflicting, and suggest many questions to be settled.

Experiments in Amalgamation[2].—In an abstract from the *South African Mines*, some experiments in amalgamation on the Rand are discussed. The Frue vanner installation at Ferreira was abandoned and amalgamation plates fitted into the vanner frames, the pulp coming from the amalgamation plates being treated on these shaking plates. It was found that 76 per cent. of the fine gold caught in this way came from the black sands, which in the cyanide works yielded only about 45 per cent. of their contents.

Mr. Hebbard, manager of the Langlaagte Estate, substituted plates with a smooth side shake for ordinary stationary plates. He found that the tailings from the shaking plate assayed, for an average of ten days, 3.27 dwt., and that the tailings from ordinary plates for the same days on the same ore averaged 4.42 dwt. The whole of the running apparatus on these shaking plates was fixed underneath, out of the way of the amalgamators. An advantage is the small amount of water required.

Amalgamating Gold Ores[3].—Algernon Del Mar, in discussing plate amalgamation, comes to the following general conclusions: First, raw copper should be used only for inside and lip plates, but may be used for battery plates when the gold is free and coarse; for all other conditions electroplated silver plates are preferred; second, no chemicals should be used on silvered plates unless absolutely necessary; third, silvered plates should never be allowed to get hard, while copper plates may; fourth, the more refractory the ore the higher the stamp discharge; fifth, a coating of amalgam should always be left on the plates; and sixth, the mill man must use care and judgment and not become a mere machine.

Gold Mining at Roudny, in Bohemia.[4]—Dr. Oskar Eypert describes the mining and milling of gold ores at Roudny, Bohemia. The gold in the ore is in a finely divided free state, disseminated through quartz, dolomite, and pyrite. The ore coming from the mine is dumped over grizzlies 10 ft. long, sloping 40 deg. and spaced 1.7 in. apart. The oversize goes to jawbreakers, the product of which, together with the grizzly undersize, goes to stamp storage bins. The ore is fed into six stamp batteries by Challenge feeders. The pulp runs over amalgamation plates to a series of hydraulic

[1]*Engineering and Mining Journal*, Vol. LXXIX (1905), p. 469. Abstract of paper read before the South African Association of Engineers.
[2]New Zealand *Mines Record*, Vol. VIII (1904–5), p. 410.
[3]*Mining and Scientific Press*, Vol. XC (1905), p. 368.
[4]*Osterreichische Zeitschrift fur Berg- und Huttenwesen*, Vol. LIII (1905), pp. 83, 101.

classifiers, spitzlutte and spitzkasten, the coarse material going in each case to a Ferraris table, while the fine material goes to the next lower classifier. The tables in each case make concentrates, tailings and slimes. The tailings and slimes go to six spitzkasten, the pulp from which goes to six Ferraris tables. These tables also make concentrates, tailings and slimes. The tailings and slimes go to the leaching plant.

The stamps drop in the order of 1, 3, 5, 2, 4, with a drop of 6 in. The stamps weigh 990 lb. The screen is 30 mesh. The amalgamating plates are 12 ft. long and 5 ft. wide and have a slope of 1 in 12. From 50 to 60 per cent. of gold contents is extracted by amalgamation. The Ferraris tables are 12 ft. long, 5 ft. broad, and use 0.5 h.p. The mill receives 140 tons per day, of which 40 tons are waste rock.

Mining Practice at the Camp Bird.[1]—Stephen L. Goodale describes the method of milling the ore at the Camp Bird mine, Ouray, Colo. The ore is a highly silicious free-milling ore. The system used is as follows:

1. Ore from mine. To (2).
2. Blake breakers. Product (passing 1.5 in. ring) to (3).
3. Stamps. To (4).
4. Copper amalgamation plates. Through mercury traps to (5).
5. Frue vanners. Concentrate; tailing to (6).
6. Spitzkasten. Coarse to (7); middling and fine to (8).
7. Huntington mills. To (8).
8. Vanners. Concentrate; tailing to cyanide plant.

The stamps are 70 in number; 10 are new, with anvil blocks, 60 are old, with wooden mortar blocks. The duty of the new stamps (1050 lb.) is 3.9 tons per stamp per 24 hours, and of the old stamps (850 lb.) 3.4 tons per 24 hours. The stamps drop 8 in. 100 times per minute. The screen is 26 mesh, 29 wire. The hight of discharge is 3 in. The four 5-ft. Huntington mills have screens 30 or 35 mesh, 29 wire; they make 75 r.p.m.; the following is a sample of their grinding:

Mesh.	Feed %	Discharge %
On 40	13.0	0.4
" 50	28.2	4.4
" 60	16.3	8.5
" 80	9.7	11.5
" 109	13.2	18.6
" 120	7.2	14.4
" 150	4.7	8.8
" 200	2.9	9.1
Through 200	4.4	24.0

The vanners treating the pulp from the plates run at 190 r.p.m., have an inclination of 3.5 to 5 in. in 12 ft., and are 6 ft. long. The vanners treating the pulp from the Huntington mills run at 180 r.p.m., have an

[1] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 850.

inclination of 2.5 in., and are 6 ft. long. In the stamp mill 87 per cent. of the total gold is recovered.

The Treatment of Refractory Sulphides at the Cassilis Mine, Victoria, Australia[1].—Francis B. Stephens describes the treatment of refractory auriferous sulphides at the Cassilis mine, Victoria, Australia. The ore contains arsenical pyrites, iron pyrites, zinc blende, galena and carbonate of iron, antimony, manganese, copper and tin, in quantities usually less than 1 per cent. Gold values average 3 to 5 oz. The plant consists of a 16x9 in. jaw-breaker, self-feeders, 20 heads of 1000-lb. stamps, silvered copper plates, four 3-cone hydraulic classifiers, six Wilfley tables, and four Berdan grinding-pans. The tailings go to settling tanks for future cyanidation and the concentrates are roasted previous to chlorination. Only 20 per cent. of the total gold is obtained by amalgamation. Each five heads of the battery have a series of three conical hydraulic classifiers under a head of 20 ft. and with diameters of 12, 18 and 24 in. The product from the four topmost compartments is run on two tables, the middle product on two and the fine on one table. The galena concentrates, which are taken off the tables in a special hopper, contain a quantity of arsenical pyrites, and are re-run on another table making three products: the head containing 40 per cent. lead goes to grinding-pans; the middling, 15 to 20 per cent. lead, 15 to 20 per cent. arsenic and 8 oz. gold per ton, goes to smelters; the third product goes back with the ordinary concentrate for chlorination.

Mill Practice on the Rand[2].—G. A. Denny gives an illustrated description of the preliminary crushing and the slime treatment at the Van Ryn Gold Mines Estate, eastern Rand. The ore is dumped over a grizzly, the fine going to a bin below while the large blocks of ore are washed; the waste rock is thrown to one side, while ore proper is crushed to 4 or 5 in. cubes. The crushed ore and undersize of the grizzly go to another grizzly. The oversize of this goes to circular picking tables, where 20 to 30 per cent. is rejected as waste. Sorted ore goes to breakers, where it is reduced to 1.5 to 2 in. cubes. The mixture of fine and coarse ore goes to the mill. The pulp from the mill goes to a series of classifiers, where the sand is separated from the slime. The slime, containing 3 per cent. dry slime, 97 per cent. water, goes to a conical vat. The slime discharged from the vat contains 30 per cent. dry slime and 70 per cent. water. From here the slime goes to a series of eight conical vats, where the per cent. of water is still further reduced. From there it goes to a second series of vats, being joined on the way by a weak cyanide solution. The slime runs 0.08 oz. gold per ton; the extraction is 73 per cent.; the working costs are 10c. per ton of ore milled.

[1]*Engineering and Mining Journal*, July 29, 1905, p. 158. Abstracted from *Transactions* Institution of Mining and Metallurgy, May 18, 1905.
[2]*Mining Magazine*, Vol. XI (1905), p. 401

Milling in Gilpin County, Colorado[1].—Brief descriptions of several mills in Gilpin county, Colo., are given. The ores of this district do not exceed $4 per ton and the average value of the concentrates does not exceed $13 per ton. The milling ore is a mixture of feldspar and quartz, carrying pyrite and a small amount of chalcopyrite. The Boston mill, operated by the Gregory-Boston Mining Company, contains eighty 1000-℔. stamps, which drop 95 times per minute, using 24-mesh screen and crushing 3.5 to 4 tons per stamp per 24 hours. The pulp from stamps passes over 4.5x16 ft. copper plates, through spitzkasten to Woodbury tables. The mill is the most advanced of any in this county. The camp is non-progressive.

St. David's Gold Mine, North Wales[2].—L. H. L. Huddart describes the mill treatment of the ore at the St. David's gold mine, North Wales. The ore is a free-milling quartz with small amounts of sulphides, such as pyrite and chalcopyrite. The ore is hand-picked at the mine into rich ore and poor ore. The rich ore is given a preliminary crushing in Blake breakers (to ½ in. cubes) and is fed into Britten pans. These pans are good amalgamators but are slow. The poorer ore goes to the stamp-mill. Here it is passed over grizzlies spaced 2 in.; the oversize goes to a Blake-Marsden breaker (10x7 in.). The crushed ore, together with the undersize from the grizzlies, goes to bins from which it is fed by Challenge feeders to 50 stamps. The pulp from stamps passes over amalgamating plates to Rose concentrators, the concentrates from which are sent to the copper smelter. The stamps are of the California type. They weigh 1250 ℔. and make 94 drops per minute. The screens are 30-mesh Russia iron, needle slot. The plates are 5x3 ft. with a slope of 1.5 in. in 1 ft. The Rose concentrators are galvanized steel boxes set at an angle, in the bottom of which are blankets covered by coarse wire screens. In June, 1904, 11.36 oz. of fine gold was caught in the three Rose concentrators. The total cost of milling, not including wear and tear, is 15 to 16c. per ton.

Notes on Crushing of Metalliferous Ores in the Stamp Battery in Africa[3].—F. C. Roberts describes the details of gold milling as practiced in South Africa in general. He states that the usual adjustments of the mill are as follows: Order of drop 1, 3, 5, 2, 4; number of drops per minute, 96; hight of drop, 8 in.; weight of stamps, 1250 ℔., some 1465 ℔.; screen 20 to 28 mesh, heavy wire cloth. Mortars are furnished with false bottoms for compensating the wear of dies. He gives the sequence of treatment in a brief summary as follows: (1) sizing, (2) sorting, (3) crushing, (4) milling, (5) amalgamation, (6) concentration, (7) classification (sand), (8) classification (slime), (9) cyanidation (sand), (10)

[1]*Mining and Scientific Press*, Vol. XCI (1905), p. 343.
[2]*Trans.* of the Institution of Mining and Metallurgy, Vol. XIV (1904–5), p. 199.
[3]*Mining and Scientific Press*, Vol. XC (1905), pp. 10, 72, 91, 105, 119.

cyanidation (slime). The author also gives the cost in detail of erecting a 20-stamp mill.

Ore Washing at Cripple Creek[1].—The descriptions of several ore-washing plants at Cripple Creek, Colo., are given. The values in the ore are usually associated with the soft material, while the hard rock of the vein is almost destitute of value. The system of washing at the Elkton mine, which may be taken as an example of the practices, is as follows:

1. Ore from mine. To ore house, to (2).
2. Grizzly, wire screen 1 in. holes. Oversize to (4); undersize to (3).
3. Bins. To elevator to (13).
4. Bins. Through chute sprayed with water to (5).
5. Sorting table 8 ft. long, men picking. Waste thrown in cars trammed to dump; ore to (6).
6. Blake breaker. Product by elevator to (7).
7. Revolving screen; 40 in. x 12 ft.; perforations $\frac{7}{8}$ in. at entrance, $1\frac{1}{8}$ at middle and $1\frac{3}{8}$ at discharge end; 8 to 12 r.p.m. Undersize to bin, to (13); oversize to (8).
8. Screen drum with a spiral screw, half under water. Slimes to (9); washed ore to (10).
9. To settling tank, drawn off, dried, sacked and shipped.
10. Picking belt with men picking ore from waste. Waste to (11); ore to (12).
11. Hopper to cars to dump.
12. Bins to breaker. Product crushed to 1 in. To (13).
13. Automatic samplers. Sample assayed; rejected ore to loading bins, to smelter.

Most of the ore of this district costs about $10 to $15 per ton by the time it is ready for the mill or smelter.

Mining Operations in the Pearl Camp, Idaho[2].—A description is given of the gold milling and cyaniding at the plant of the Lincoln Mining Company, Pearl, Idaho. The ore is a mixture of altered granite and quartz, heavy with pyrite, and carries some lead and zinc carbonate and sulphides. The ore is very slimy in character, and runs $5 to $15 gold and silver, of which 90 per cent. is gold. The ore is delivered to the mill by a 450-ft. conveyor system. It is fed through grizzlies to a Gates breaker. The crushed material goes to a 20-mesh trommel; the oversize goes to a Chilean mill (10-mesh screen) and the undersize to Cornish rolls. The product from the rolls, crushed to 10 mesh, goes to the Chilean mill. The pulp from the mill passes over two 10-ft. amalgamation plates where 15 per cent. of the value is saved. The pulp from the plates is pumped to wooden tanks, from which it is fed into Eldredge slime cones.

[1] *Mining and Scientific Press*, Vol. XC (1905), p. 102.
[2] *Mining Reporter*, Vol. LII (1905), p. 498.

The sand goes to the leaching tanks and slime to settling ponds. The slime from the cone runs about 20c. gold and 69c. silver per ton. The slime-separating cone consists of a cone 7 ft. long, with diameter of 6 ft. at the top and with sides sloping 60 deg. The pulp is fed by a pipe dipping 16 in. below the rim; the slimes overflow the edge into a launder; this is aided by an upward flow of clear water sprayed from an arc at the apex of the cone. The sands settle through the annular ring around the arc and are drawn off into the leaching tank below. The four slime-separating cones have a daily capacity of 30 tons each.

Ore Milling at Kalgoorlie[1].—Harry J. Brooke discusses the dry crushing mill of the Associated Northern Blocks, Kalgoorlie. The ore is a complexed sulpho-telluride. At the mill the ore is dumped over grizzlies to a No. 5 Gates gyratory breaker. The ore, crushed to 2½ in., is distributed by a belt-conveyor into a bin, from which it is fed by automatic bump-feeders into 3 No. 5 Krupp ball-mills. These mills have woven wire screens 25 mesh, 26-gage wire; their capacity with 2000 lb. of balls is 40 tons per 24 hours; 53 per cent. of crushed ore from the mill passes 120 mesh. The steel balls wear off 3 to 3.5 oz. per ton of ore crushed. The crushed ore is then conveyed to a bin, from which it is fed into five Merton roasting furnaces. The cooled ore from the roasting furnaces goes to six 5-ft. grinding pans. Ninety-two per cent. of pulp from the pans passes a 120-mesh screen. Forty per cent. of gold is recovered by amalgamation, at a cost of 0.2 oz. of mercury per ton. The pulp goes to settlers, from which the thickened pulp goes to cyanide agitators. The total cost of treatment at mill is $3.07 per ton.

Gold Mining Districts of Western Australia[2].—W. T. Saunders describes briefly the mill practice at the following mines: Ivanhoe; Lake View Consols; Oroya Brownhill; Golden Horseshoe; Kalgurlie; Associated Mine; Associated Northern Mine; Great Boulder Proprietary Mine; Great Boulder Main Reef Mine; and Great Boulder Perseverance Mine. The first four crush wet in stamp-mills, amalgamate, concentrate and cyanide. The remainder treat the ore as follows: Dry crushing in Krupp ball-mills or Griffin mills, roasting in Edwards or Merton furnaces, pan grinding, classification and concentration, then cyaniding.

The Cerro Prieto Mine and its Equipment.[3]—A description is given of the power plant, amalgamation plant and cyanide plant put up at Cerro Prieto mine, Sonora, Mexico, to treat ore from a vein 75 ft. wide running from $2 to $6 per ton. The ore as it comes from the mine will be fed over a grizzly to a No. 5 Allis Chalmers breaker (18x10 in.). The ore broken to 6 and 8 in. goes to two Gates breakers, where it is broken to 1.25 in. size. From the breaker house the entire product will be belt-conveyed to

[1]*Engineering and Mining Journal*, Vol. LXXX (1905), p. 4.
[2]*Trans.* of the Institution of Mining Engineers, Vol. XXVIII (1905), p. 585
[3]*Mining Reporter*, Vol. LI (1905), p. 599.

the stamp-mill bins, from which it will be fed by Challenge feeders into 120 stamps. From the stamp-mill the pulp will pass over 12-ft. amalgamating plates to draining tanks in the cyanide plant. Details are: Weight of stamps, 1050 ℔.; battery screen, 35 mesh; weight of mortars, 10,000 ℔. each; stamps make 106 r.p.m. and drop 6 to 7 in. Every 20 stamps will be driven direct by a 50 h.p. induction motor.

Plant of the Palmarejo and Mexican Gold Field Company, Ltd., in Chihuahua, Mexico[1].—The ore is a silicious matrix, with disseminated pyrite, some black manganese oxide and calcite, carrying native silver and argentite. The system of milling is as follows:

1. Ore (6 per cent. moisture) from mine to storage bin. To (2).
2. Grizzlies (3.5x10 ft., ½ in. spaces). Oversize to (3); undersize to (4).
3. Blake breaker, 7x10 in., 250 r.p.m., crushing to 2 in. Product to (4).
4. Bins. Trammed to (5).
5. Challenge ore-feeders. To (6).
6. 50 stamps with 20-mesh screen. To (7).
7. Wilfley tables. Concentrate saved; pulp to (8).
8. Bucket elevator to sand tanks. Sand to (9); slime to (10).
9. Leaching tanks.
10. Slimes put to drying patio where they are mixed with coarse sand and go to leaching tanks. Details are: Stamps, 850 ℔.; drop, 6 to 7 in.; number of drops per minute, 100; screen, 20 mesh, brass wire; hight of discharge, 2 in.; duty 2.75 to 3.25 tons per stamp per 24 hours. The Wilfley tables have ⅞ in. stroke and 215 per minute. They save 0.76 per cent. by weight of the ore, which contains 18.28 per cent. of the gold and 17.98 per cent. of the silver in the ore.

Present Stage of Metallurgy on the Rand.[2]—G. A. and H. S. Denny describe fully the present practice of gold milling on the Rand, and offer an alternative method of treatment as an improvement on the present method. The authors give, in tabular form, the extraction, cost of extraction, tonnage, etc., of concentrates, sands and slimes in some nine companies, and show how these results might be improved upon. The present treatment of the ore is as follows:

1. Single breaking of ore, after passing over sorting tables, to 2 or 3-in. cubes.
2. Milling with 600- to 1200-mesh screens.
3. Outside plate amalgamation.
4. Spitzlutten separation of from 5 to 15 per cent. of heavier pulp, with subsequent cyanide treatment of separated products, occupying from 14 to 40 days.
5. Subsequent separation of sands, representing 55 to 70 per cent. of

[1] *The Engineering and Mining Journal*, Vol. LXXX (1905), p. 297. Abstracted from the *Trans.* of the American Institute of Mining Engineers, May 1905.
[2] *Mining Magazine*, Vol. XII (1905) p. 173

pulp with treatment of 5 to 10 days; simultaneous separation of 22 to 35 per cent. of slimes and treatment by decantation.

The authors' alternative method is as follows:

1. Stage breaking down to $\frac{3}{4}$ to $1\frac{1}{2}$-in. cubes.

2. Milling with a coarse mesh of 200 holes to the square inch, followed by plate amalgamation.

3. Classification of the coarse product in the mill pulp in spitzlutten.

4. Regrinding of the coarse products (everything over 60 mesh) in tube-mills or other suitable grinding machine. If pans were used amalgamation in pans would be resorted to.

5. Passing the reground products over a further set of amalgamating plates.

6. Automatic agitation of the whole pulp in cyanide solution. Practically an all-sliming process.

7. Filter pressing the pulp after agitation.

They claim for this method lower initial cost, lower working cost and higher extraction. The authors also state that the grinding-pan, such as the Wheeler type, is to be preferred to the tube-mill for fine grinding.

The authors believe that the days of the stamp-mill are numbered with respect to the future equipment of a large number of the deep-level mines on the Rand. The reason they give is that the breakers overlap the stamp on one side and the grinding plant on the other, so that there is no room for the stamp; further, the stamp is an inefficient crusher and has high first cost, also a high working cost.

Reduction Plant and Process at the Oroya-Brownhill Mines[1].—In an abstract of the *Journal* of the Chamber of Mines, Kalgoorlie, West Australia, Robert Allen describes the plant of the Oroya-Brownhill mines. The ores from the three mines are similar in character—schistose—and crush freely; also they are free from minerals harmful to cyanide process. Tellurides of gold and silver with auriferous iron pyrites form the bulk of the concentrates. The ore at the mines is passed over grizzlies, spaced 1.5 in.; the oversize is crushed in No. 6 Blake breakers and goes with the undersize to the mill at the Brownhill mine. At the mill the ore is fed into a battery of fifty stamps, each of 1100 lb. The pulp from the battery goes to a pair of conical spitzkasten which make coarse sand, fine sand and slime. The coarse sand goes to five Wheeler grinding pans, the product from which goes to a series of spitzkasten which separates into coarse and fine. The coarse goes to coarse tables, and the fine, together with the fine from the battery spitzkasten, goes to fine tables. These tables make two concentrate products and tailings. The latter runs to conical spitzkasten making coarse and fine. The coarse goes to six flint mills; the fine goes to six

[1]*Mining and Scientific Press*, Vol. XCI (1905), p. 366.

tables, making a third concentrate, the tailings going through a series of spitzkasten and condensers to the leaching department.

The mortars of the stamp batteries are of Homestake pattern. The steel consumption of shoes and dies in batteries are respectively 4.36 and 4.56 oz. per ton crushed. The stamps drop 7¾ in. 108 times a minute, and, with a 10-mesh wire screen, have a duty of 6.48 tons per stamp per 24 hours. The grinding pans run at 51 revolutions per minute. A set of shoes and dies lasts three months. The tube-mills are 13 ft. 7 in. long, 3 ft. 8 in. diameter. They are lined with chilled cast iron. Each mill wears 470 ℔. of lining per month. The mills are driven at 32 r.p.m. and use 25 h.p. each. About 238 tons run through each mill per day and each mill actually fine grinds about 27 tons per day, equivalent to 160 tons per day for the tube-mill plant. The concentrates from the table are roasted and ground in pans with mercury, where 30 per cent. of their value is recovered; the tailings are leached with cyanide.

Hydraulicking.

The Washing of Gold-Bearing Sands in Sluices[1].—L. St. Rainer gives a mathematical discussion of the mechanics and behavior of particles of gold and sand moving under the influence of water and gravity in sluice-boxes. He arrives at various formulæ for the action of particles of gold and sand on sluices with angle-iron riffles, with cocoa matting and other forms. He comes to the conclusion that, if sluices are properly constructed, a length of from 8 to 10 ft. is sufficient to catch 999 parts of gold in a thousand.

Hydraulic Mining[2].—Richard L. Grider describes briefly the best methods of working gravel deposits by hydraulic power. He describes methods of building pipe-lines, sluice-boxes and hydraulic elevators. He gives the following points to be observed in hydraulic mining; Waste gates should be put in on ditches and flumes every quarter or half mile, as a protection against floods. In the pressure-box the water should stand not less than 4 ft. deep over the entrance of the pipe, to prevent the admission of air. The pipe-line is best constructed of iron, coated with a mixture of coal tar and asphalt, and with as few elbows and depressions as possible. The best type of giant is double-jointed giant, with king-bolt bearing and with deflector. The bed-piece of the giant should be properly staked and braced with wooden stakes. In bed-rock sluices, iron riffles catch the gold best, but in case of sluices over 200 ft. the less expensive wooden riffles are to be preferred. These latter are spaced 1 to 3 in. apart at the head of the sluice, decreasing toward the dump and with the heart side of the block up stream. The spreading of the dump is facilitated by means of branch- or

[1]*Osterreichische Zeitschrift für Berg und Huttenwesen*, Vol. LIII (1905), pp. 55, 69.
[2]*Bulletin* of the Colorado School of Mines, Vol. II, No. 4, p. 18.

Y-sluices. In clean-up the bed is washed with the pipe into the race and the race into the flume. When the race is clean the lining is ripped off and the riffles taken out and thoroughly washed. The material on the bottom of the flume is loosened with shovels and washed down the flume. The mercury and amalgam are dragged along behind and are scooped up.

The author states that ordinarily 90 to 98 per cent. of the gold should be saved, and with coarse and heavy gold 98 to 100 per cent. should be saved.

Gold Deposits of the Klondike[1].—J. M. Bel describes briefly the methods of working gold deposits in Yukon, Canada. The first method is tunneling, a steam trident being used to thaw the frozen ground. The second method is to use a scraper on the surface; the scraper is drawn by a cable from a steam engine, and helped to its work by men. The third method is hydraulicking. The water has to be pumped by centrifugal pumps to the required head. The character of the material is not argillaceous but sandy, and the gold is round, so that comparatively short sluices are necessary, 60 to 90 ft. The riffles are of wood, and very crude. The cost of working is from $3 to $4 per cubic yard.

Methods and Costs of Gravel and Placer Mining in Alaska[2].—Chester Wells Purington describes thoroughly methods and costs of gravel and placer mining in Alaska. The following is a portion of a table giving the average capacity and cost of gold gravel mining in the different provinces of Alaska.

AVERAGE CAPACITY AND COST OF GOLD GRAVEL MINING IN NORTHWEST AMERICA.

	Hydraulicking. No pumping of water.	Hydraulicking with use of hydraulic elevator.	Open cut; shoveling into sluices. No pumping.	Open cut; horse scraping.	Open cut; steam shovel, track and incline to sluice.	Dredging.	Hydraulicking by means of pumped water.	Booming with self-dumping water gate.
South Coast Province.								
No. operations considered	6	6	6	....	....	....	....	....
Capacity cu. yds. in 24 hrs.	833	350	54	....	....	....	....	....
Thickness of deposit, feet.	30.3	25	5.6	....	....	....	....	....
Thickness of gravel worked, feet.	30.3	25	3.7	....	....	....	....	....
Cost per cu. yd. (*a*)	$ 0.20	$ 0.31	$ 2.01	....	....	....	....	....
Interior Province.								
No. operations considered	13	....	20	....	....	....	4	....
Capacity cu. yds. in 24 hrs.	1049	....	63	105	800	1062	830	250
Thickness of deposit, feet.	37.4	....	8.6	20	22	35	33	75
Thickness of gravel worked, feet.	37.4	....	3.5	(*b*)10	22	35	33	(*b*)60
Cost per cu. yd. (*a*)	$ 0.238	....	$ 2.39	$ 0.60	$ 1.46	$ 0.49	$ 0.65	$ 0.07
Seward Peninsula.								
No. operations considered	....	4	10	5	....	....	3	....
Capacity cu. yds. in 24 hrs.	....	658	145	200	1000	700	250	....
Thickness of deposit, feet.	....	12	6.6	5	30	8	23	....
Thickness of gravel worked, feet.	....	12	3.3	5	27	8	23	....
Cost per cu. yd. (*a*)	....	$ 0.89	$ 1.87	$ 0.46	$ 0.52	$ 0.43	$ 0.93	....

(*a*) Lost time, prices paid for mining property and the cost of equipment other than that relating to actual mining are not taken into account. (*b*) Muck and top gravel.

[1] *Bulletin* de la Société de l'Industrie Minerale, Series IV, Vol. IV (1905), p. 275.
[2] *Bulletin* No. 263, U S. Geological Survey.

The following is a selected portion of a table giving the duty of a miner's inch in the northern placers. (Duty of a miner's inch is the quantity of material moved by 1 inch in 24 hours.)

Locality.	Hight of bank	Grade inches in 12ft.	Miner's inches of water.	Cu. yds. duty.	Sluice.		Riffles.	Remarks.
					Width.	Depth.		
	Feet				Inches	Inches		
Birch Creek, Atlin	25	5½	1200	½	30	30	Blocks	Heavy Stones
Spruce Creek "	29	5	1200	1	40	36	"	" "
McKee Creek "	40	8	700	1½	32	32	Angle iron&blks	" "
Gold Creek, Juneau	200	4½	4000	2	72	60	Blocks	" "
McKee Creek, Atlin	85	8	1200	3	32	40	Angleiron&blks	" "
Bonanza Creek, Dawson	25	12	125	4	20	16	Ironshod pole	Hillside small frozen gravel.
" " "	20	12	250	5	24	20	Blocks	Small round gravel, frozen.
" " "	35	12	200	6½	24	24	"	Small gravel, frozen.
Last Chance "	46	12	230	7	24	18	Blks & i.s. poles	" " "
Bonanza Creek "	75	12	266	8	24	30	Blocks	" " "
" " "	25	11	150	10	24	24	"	Small round gravel, part tailings.

Many other tables of interest and many photographs and drawings are given, concerning which the reader is referred to the original. The publication is a valuable treatise on the subject of hydraulic mining in general, though confined in its treatment to Alaska.

Hydraulic Mining in Humboldt County, California[1].—H. DeC. Richards describes briefly the hydraulic mining of the Orleans Bar Gold Mining Company on the Klamath river. A gravel bank 100 ft. high is attacked with a 7-in. giant under 225 ft. head, water being brought 14 miles in flumes. A transit survey of the bank is made after each clean-up the first of the month. The average total cost of washing in 1903-4 was 2.483c. per cubic yard; 6.92 acres were washed during that year, with an average flow of 226 miners' inches of 1.5 cu. ft. per minute.

The Dividend Gold Saver[2].—A gold saver used to save fine gold consists of plates with holes in them placed upon plush or matting. The approaches to the holes are so sloped and graded as to produce eddy currents which precipitate fine gold. It is in use on the Rising Sun dredge, near Cromwell, Otago, and is said to save 3 per cent. more of the fine gold than ordinary expanded metal and matting did.

Hydraulic Mining on Banner Placer, Colorado[3].—Kirby Thomas describes the methods of working the placer deposits on Clear Creek, Colo. The Banner placer is worked by three hydraulic giants and an automatic boom. The giants have a capacity of 1200 cu. yd. each, with a three-inch nozzle and 2000 yards with a six-inch nozzle. The cost of moving the gravel with the giants for operation and maintenance is about 2.5c. a cubic yard. The automatic boom is adapted to utilize the surplus water and to carry the gravel away from the first sluices. Vibrating riffles of 2.5-in. angle steel are used. An auxiliary mill, supplied with Acme tables, con-

[1]*Engineering and Mining Journal*, Vol. LXXIX (1905), p. 362.
[2]The New Zealand *Mines Record*, Vol. VIII (1904-5), p. 292.
[3]*The Mining World*, Vol. XXIII (1905), p. 306.

centrates the black sands into a product containing $25 to $50 in gold and other minerals.

Platinum in the Cariboo[1].—This is a brief description of hydraulic mining by the Consolidated Cariboo Hydraulic Mining Company in British Columbia. The gravel carries about 15c. per yard. It is worked by six No. 8 giants and values are collected in 3400 ft. of partly steel-riffled sluice. Undercurrents for recovery of flour quicksilver and the heavy metals are being erected at the dump. An analysis of the heavy concentrates after clean-up gave the following:

	Oz. per ton.	Gross value per ton.
Gold	95	$1900
Silver	180	90
Platinum	64	832
Palladium	61.4	1769
Osmiridium	42.	1386
Copper	10.5	17

Gold Dredging.

Gold Dredging and Prospecting[2].—Robert H. Postlethwaite discusses gold dredging and prospecting land for dredging from a practical point of view. He quotes the working expenses of nine dredges in Oroville, Cal., of which the following are examples: 3½-ft. dredge, period of one complete year. Total yards handled, 290,616; costs per yard as follows: Expenses account, 0.059c.; repairs and supplies, 1.293c.; labor, 3c.; power, 0.63c.; total, 4.982c. Directors' fees and taxes not included.

Five-foot dredge; one month run. Bank measure handled, 50,760 cu. yd. Cost of all labor, $1042; power, $422; cost of all supplies, $394; total costs, $1858. This is equivalent to 3.66c. per yard. Bullion recovered, $9600.

In another case, with a four-foot dredge working in a very hard compact formation, the cost goes as high as 8.7c. per cubic yard.

Peck's Centrifugal Elevator[3].—W. Peck describes his centrifugal elevator used to elevate tailings from dredges. The elevator consists of a cam-shaped steel casting with a disk of mild steel plate on either side. The two beater surfaces are faced with renewable strips of hard, tough steel. The beater revolves for the best work at 240 r.p.m. The cost of upkeep is about $2.50 per week. It is most efficient on small stone, and less so on sands and silt.

Gold Ships and their Cargoes[4].—Alexander Del Mar discusses the cost of operating and the profits of gold dredging, and gives several photographs with descriptions of dredges of the chain-bucket type. He gives a case of a

[1] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 332.
[2] *Mining Magazine*, Vol. XI (1905), p. 5.
[3] *Trans.* of the Australasian Institute of Mining Engineers, Vol. X (1905), p. 265.
[4] *Engineering Magazine*, Vol. XXIX (1905), p. 481.

thirty-two, 5-ft. bucket dredge costing $50,000, with working expenses of $16,800 per annum, which excavated 500,000 cu. yd., averaging 20c. per yard, in a year, and gave a net profit of $83,000, or 128 per cent.

Dredging[1].—J. P. Hutchins discusses the development, practice and improvements of dredges. He advises the use of a prospecting dredge to prove the ground and to forecast costs where a large expenditure is to be made. He advises the use of tumblers on the dredges, with more numerous faces, the advantage being less friction and less surging. He prefers trommels to shaking screens, as they use less power, give less trouble and give cleaner screening. Of gold-saving devices the hydraulic mining undercurrent type and the mercury riffle with drops are the best. Woven matting for catching rusty gold is very good. "Save-alls" sometimes save 10 per cent. of the gold. Trough belt-stacker is superior to the bucket-stacker. To keep the tailings off virgin ground the distance from tailings to side of cut should be twice the depth to which the dredge is digging.

Improvements in Gold Dredges[2].—George E. Walsh states that the tendency is to build larger and more powerful dredges. The previous largest dredges were those with 65 buckets with a capacity of 6 cu. ft. each; now they are increased to 75 buckets with a capacity of 7 cu. ft. each, handling upward of 100,000 cu. yd. per month. The use of cheap electric power has given great headway to the development of the dredge.

Dredging and Dredging Appliances[3].—Brysson Cunningham describes, with profuse illustrations, several dredges of the grab-bucket type and of the centrifugal suction type.

European and American Dredges[4].—Frank C. Perkins describes, with photographs and drawings, the different types of chain-bucket, grab and dipper dredges.

Gold Dredging in California.[5]—A. G. Hillen describes gold dredging as practiced in California. The ground dredged near Oroville, Butte county, Cal., is in general deposits of gravel and alluvial sands, with a depth of 30 to 50 ft. down to bed rock. The bed rock consists of a fine lava ash, and is easily cleaned of both gravel and gold. In some cases the gravel is cemented together; it is then worked by drilling holes 50 ft. apart and shattering the cement gravel with a No 2 giant powder. The gold is in very fine condition, and few nuggets are found. The average value is about 15 c. per cu. yd. The cost of working ranges from 3.5c. to 8c. per cu. yd. An example of the distribution of expenses in a 5-ft. Bucyrus dredge which handled 895,506 cu. yd. in a year is as follows: Pay roll, $0.01136; power, 0.01347; dredge supplies, 0.00238; taxes and insurance, 0.00185; sundry expenses, 0.00348; maintainance and repairs, 0.00383; total, $0.03637.

[1]*Engineering and Mining Journal*, Vol. LXXX (1905), pp. 49, 102.
[2]*Ibid.*, Vol. LXXX (1905), p. 246.
[3]*Cassier's Magazine*, Vol. XXIX (1905), p. 132
[4]*Mining Reporter*, Vol. LI (1905), p. 373.
[5]*Mining World*, Vol. XXIII (1905), p. 118.

The author describes several dredges. One of the Risdon type has a hull 94 ft. long, 40 ft. wide and 7 ft deep. It weighs 225 tons and took 5½ months to erect. The front gantry is of steel channel irons, while the stern gantry is of 14x14 in. Oregon pine. The ladder is arranged to dig 35 ft. below the water and to stack the tailings to a hight of 40 ft. above the water line. The gold-saving apparatus consists of a revolving screen 4.5 ft. diameter and 30 ft. long. with gold-saving tables on each side. The water is supplied by centrifugal pumps. The whole dredge is worked by induction motors, three-phase 7200 alternations per minute. Power costs 1.5c. per kw.-hour and amounts to $700 per month. The crew consists of three shifts of two men each, with one dredge captain. The dredge actually handles 150 yd. per hour. Most of the dredges of this district are driven by electric power, as expensive fuel makes steam power high.

Dredging at Oroville.[1]—L. J. Hohl discusses the running time and the cause of stoppages in dredges. In the case of a Bucyrus dredge the bucket line, the upper tumbler, and high water caused the greater percentage of the stoppage. Taken in a year of 365 days of 24 hours each, this dredge averaged 16 hours, 56 min. per day. For a Risdon dredge the average running time amounted to 16 hours, 39 min. out of 24 hours. A fair average yardage would be about as follows:

	Bucket Capacity.	Yd. per month
Risdon Dredge........	3 cu. ft.	25,000 to 39,000
" "	5 " "	35,000 to 45,000
Bucyrus Dredge.......	3 " "	35,000 to 45,000
" "	5 " "	50,000 to 65,000

The following statement, giving the operating expenses per cubic yard of material, was compiled from data given by operators in the Oroville district.

Power	1.06	1.20	1.15	1.61	1.77
Repairs....................	2.86	3.03	3.46	2.97	3.80
Labor	1.64	1.82	1.85	2.33	2.05
General expenses...........	0.64	0.67	1.23	1.28	0.73
Totals, cents	6.20	6.72	7.69	8.19	8.35

A dredge should last 12 to 15 years, during which time the tumblers, ladder-rollers, buckets, shaking tables, etc., will have to be replaced over and over again.

Dredging in California.[2]—A description is given of gravel mining in California, with particular reference to dredging. The different kinds of gravel mining are: First, ordinary hydraulic mining, requiring water under pressure and a good dump, with gravel easily removable. Second, drift

[1] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 895.
[2] *Mining and Scientific Press*, Vol. XC (1905), pp. 125, 141, 160, 178; abstracted from Bulletin No. 36 State Mining Bureau of California.

mining in layers of gravel deposits over-capped with lava or other material of such great depth that only underground mining is possible. Third, working river bars with hydraulic elevators, where dumping facilities are lacking, and where the bed rock is too hard for a dredge and where the water is not in too great excess. Fourth, where there is an excess of water and a soft bed-rock which can be worked only by dredge. The gold dredge consists of a hull, a digger, screen, sluice-table and sluice-boxes, a stacker, a pump, amalgamator and often a sand-pump with lines or lines and spuds. The hulls are of wood and vary from 30 to 40 ft. in width, and are from 80 to 120 ft. in length, and from 7 to 9 ft. in depth. The gantries are generally of wood, sometimes of steel plates. The forward one supports the digger ladder, and the stern one supports the tailings sluice-conveyor ladder and spuds, if used. The digger consists of an iron or steel ladder frame, upon which travels a continuous chain of buckets. These buckets may be close set, one to every link, as in the Bucyrus, or open set, one bucket to every other link, as in the Risdon dredge. The former gives better efficiency in loose gravel where there are no large boulders, but the latter last longer where there are many boulders. The buckets vary in capacity from 3 cu. ft. to 8.5 cu. ft. and a dredge is being built to have a capacity of 13 cu. ft. per bucket.

Both spuds and head lines are used to keep the dredge up to its work. Of the two methods, the spuds seem to be a little better for dredging, especially in deep ground. The two forms of stackers used are the endless chain bucket-conveyor and the belt-conveyor. The former lasts longer and will work at a higher grade; the latter, however, is more easily and more quickly replaced, and is less expensive. Both revolving and shaking screens are used; the former work the best, especially where the gravel is lumpy. The size of openings depends on the size and shape of the gold particles, Revolving screens vary in diameter from 3.5 ft. and in length up to 24 ft. The area of a shaking screen for a 5-ft. dredger is about 750 sq. ft. Hungarian riffles with mercury are used where the gold is easily amalgamable, but where the gold is rusty or there is much platinum, cocoa matting with expanded metal is preferred. The cost of dredging varies from 2.36 c. per cu. yd. for a new dredge to 8.5c. per cu. yd.

A Large Gold Dredger.[1]—A description is given of a large gold dredger erected by the Folsom Development Company, of Folsom, Cal. The dredger is of the single-lift type with continuous buckets, flat screens and rubber belt-stacker. It is driven by a 150-h.p., variable speed, induction motor, and is provided with head line anchorage as well as spuds. The hull is 120 ft. long, 43 ft. wide and 11 ft. deep over all. The bucket chain consists of 82 buckets of about 9 cu. ft. capacity each. A bucket complete weighs 2550 ℔. The bucket chain is driven at the rate of about 45 ft.

[1] *Mining and Scientific Press*, Vol. XC (1905), p. 282.

per minute, delivering the contents of 20 buckets per minute. The gravel is delivered to flat screens arranged in tandem at 130 strokes of 4 in. per minute. The values are washed through the screens by 5000 gal. of water per minute to the mercury tables below. The coarse material goes to a rubber belt-conveyor and is stacked. The dredge will handle 125,000 to 140,000 cu. yd. per month.

The Extension Sluice in Dredging.[1]—This article discusses the use of the sluice extension to be used in place of tailings elevator. It is claimed that it leaves the land in a more level condition, but it is also stated that the leveling of dredging land is largely a matter of sentiment.

[1]*Mining and Scientific Press*, Vol. XC (1905), p. 387.

GRAPHITE.

By Edward K. Judd.

The domestic production of natural graphite has never afforded much more than one-eighth of the consumption of that material in the United States, the main demand being supplied by imports, free of duty, from Ceylon, India, Germany and Canada; the artificial production, also, is becoming an important source of graphite.

Natural Graphite.—The natural graphite mined in the United States may be conveniently classified as crystalline or amorphous, the distinction being based upon mode of occurrence as well as upon adaptability to particular uses. The former is more nearly pure carbon, and occurs in crystalline flakes or masses, which can, with comparative ease, be concentrated and purified so as to yield a product suitable for the manufacture of refractory ware, lubricants, pencils and polishes, uses for which a high-grade product is indispensable. Amorphous is a general term covering all other natural graphite, occurring in intimate association with argillaceous and other impurities, from which it can not readily be separated and is therefore confined to the less exacting manufacture of paints and foundry facings.

The leading source of crystalline graphite is the Adirondack region of New York, where the Dixon Crucible Company is the principal operator. Chester county, Pennsylvania, contributes some crystalline graphite. The supply of amorphous graphite comes mainly from Rhode Island, Michigan and Wisconsin, with smaller amounts from Colorado, Ohio, North Carolina and Georgia, that from the last State being of particularly crude nature, applicable only as a coloring agent for artificial fertilizers.

Artificial Graphite.—The International Acheson Graphite Company of Niagara Falls, N. Y., with the recent completion of a large extension equipped with the most modern electrical and mechanical appliances and machinery, has doubled the capacity of its plant for making graphite in the electric furnace, and has closed a contract with the Niagara Falls Power Company for another 1000 h.p. of electrical energy in addition to the 1000 h.p. previously used. The present commercial importance of artificial graphite is shown by the table following, in which it is seen that the amount of graphite made in 1905 exceeded the amount of crystalline graphite mined in that year, and that its average value per pound was nearly double that of the best natural product. Acheson graphite is largely used in the manufacture of metal protective paints, dry batteries, stove polish,

packing and as a lubricant. The electro-chemical processes also consume a good deal of it.

STATISTICS OF GRAPHITE IN THE UNITED STATES.

Year.	Refined Crystalline Graphite.						Amorphous Graphite. Production.		Artificial Graphite. Production.	
	Production.		Imports.		Consumption (c)					
	Pounds.	Value (b)	Pounds.	Value.	Pounds.	Value.	Tons 2000 lb.	Value.	Pounds.	Value.
1896	405,006	$ 18,225	33,824,000	$ 437,189	34,229,006	$ 455,414	574	$ 3,850	....	....
1897	993,138	44,691	19,113,920	270,952	20,107,058	315,643	1,200	11,400	162,382	$10,149
1898	1,647,679	82,385	30,199,680	743,820	31,847,359	826,205	1,200	11,400	185,647	11,603
1899	3,632,608	145,304	41,586,000	1,990,649	45,218,608	2,135,953	1,030	8,240	405,870	32,475
1900	4,103,052	164,122	32,298,560	1,389,117	36,401,612	1,553,239	1,045	8,640	860,750	68,860
1901	3,967,612	135,914	32,029,760	895,010	36,997,372	1,067,921	809	31,800	2,500,000	119,000
1902	4,176,824	153,147	40,857,600	1,168,554	45,034,424	1,322,401	4,739	55,964	2,358,828	110,700
1903	4,525,700	164,247	32,012,000	1,207,700	36,537,700	1,371,947	16,591	71,384	2,620,000	178,670
1904	4,357,927	162,332	25,350,000	905,581	29,707,927	1,067,913	19,115	102,925	3,248,000	217,790
1905	4,260,656	170,426	34,914,611	983,034	39,175,267	1,153,460	......	....	4,595,500	313,979

The exports of graphite from the United States were valued at $834 in 1901, $365 in 1902, $4,220 in 1903, $8,455 in 1904, and $91 in 1905. (b) Nominal. (c) Neglecting the small re-exports of foreign produce.

Graphite Mining in the United States.

California.—Graphite is reported from seven counties of California. The most important results have been from the Skinner mine, in Sonoma county, which has been in operation since 1894, and has produced about $9000 worth of mineral paint. The development works are at present, however, inaccessible and there is no outcropping of graphite.

The Western Graphite Company, of San Francisco, operates a mine in Mendocino county, about 15 miles east of Point Arena. The graphite occurs in a blanket formation; it is mined by quarrying, then washed and puddled to free it from quartz inclusions, and later refined in San Francisco. It is sold for use as a paint, as a lubricant, and for foundry facings.

New York. (By D. H. Newland.)—There was a marked advance in the Adirondack graphite industry during 1905. The output was larger than ever before, owing to the additional supply from companies which became active producers for the first time; unusual interest has been shown in prospecting for new deposits, in some cases leading to promising developments. While productive operations have so far been confined to the region bordering Lake George and southern Lake Champlain, it may be noted that the distribution of graphite is widespread throughout the metamorphosed sediments of the Adirondacks.

The American Graphite Company, a branch of the Joseph Dixon Crucible Company, continued to work its mine at Graphite, near Hague. The mine is remarkable for its size and the excellence of its product. The graphite occurs in the form of flakes distributed through quartzite. The rock is crushed at the mines and concentrated by means of buddles and air jigs to an average of about 70 per cent. graphite. The concentrates undergo further treatment in the mill at Ticonderoga; the final products consist of high-grade flake graphite and residue. The flake graphite is used mostly

as lubricant, while the residue is mixed with imported graphite and made into crucibles.

The Columbia Graphite Company opened a mine near Rock pond in the town of Ticonderoga. In character the deposit is similar to that at Graphite, but it does not average quite as well and the graphite occurs in smaller flakes. Pyrite and pyrrhotite accompany the graphite; they are found both in veins and in granular particles distributed throughout the quartzite. Mica is also present. The company has erected a mill at Rock pond with a capacity of 3000 lb. of graphite per day. The product is hauled by wagon to Ticonderoga for shipment.

A very large area of graphitic quartzite in the same vicinity is owned by John D. Bly, who is preparing to develop the deposit.

In the western part of the town of Crown Point, the Crown Point Graphite Company has opened a bed of graphitic limestone, and has a mill in course of erection. It is intended to separate the graphite by dry crushing and passing the product over screens. The separation should offer no particular difficulty, as the graphite is loosely held in a soft matrix.

On South bay, near Whitehall, the Adirondack Graphite Company and the Champlain Graphite Company have been active during a part of the year. Extensive beds of quartzose schist are found at this locality. The graphite, owing to its finely divided condition, requires special treatment for its recovery.

Among other occurrences in the Adirondacks that give promise of future importance are those at Johnsville on the Adirondack branch of the Delaware & Hudson Railroad, where the graphite occurs in crystalline limestone. Up to the present, however, they have not been worked on a commercial scale.

Pennsylvania.—The crystalline graphite of Chester county is worked by the New Philadelphia Graphite Company, of Philadelphia, whose mine and plant are at Chester Springs, and by the Federal Graphite Company, also at Chester Springs. The former is the successor of the Philadelphia Graphite Company, and spent the larger part of the year in reconstructing its mill. The plant of the Federal company went out of commission in July.

The Columbia Graphite Company, which was recently organized in Philadelphia with a capital of $750,000, has purchased the property formerly worked for graphite, near Gabelsville, Berks county, and proposes to re-open the old mine. Several slopes have been run, and the indications are said to be sufficient to warrant further work.

Owing to the suspension of output in Pennsylvania, exact returns are not available; from producers' estimates it is probable that 445,000 lb. of graphite was produced during 1905 in this State.

Virginia.—This State has not heretofore been a producer of graphite;

recent development, however, in Albemarle and Orange counties, at the base of the Blue Ridge, has opened up promising deposits, which are to be worked by the Naylor-Bruce Graphite Company, of Charlottesville, Va.

In the Naylor and the Bruce mine, the graphite occurs among gneisses and syenites in veins dipping at 45 deg. to the east. They range from 13 in. to 8 ft. wide and are clearly defined from the foot and hanging walls by clay selvages. The graphite is dense and massive, permitting the extraction of single blocks weighing several hundred pounds. The crude ore, analyzed by Froehling & Robertson of Richmond, showed 76.28 per cent. graphitic carbon.

The operating company owns 624 acres of land on which graphite appears, and is planning at once to build a factory for refining the product.

WORLD'S PRODUCTION OF GRAPHITE.

(In metric tons.)

Year	Austria	Canada	Ceylon (d)	Germany	India	Italy	Japan	Mexico	Sweden	United States (b)	Totals
1896	35,972	126	10,463	5,248	(a)	3,148	215	620	14	184	55,356
1897	38,504	396	19,275	3,861	61	5,650	204	907	99	450	69,311
1898	33,062	1,107	78,509	4,593	61	346	346	1,857	50	824	125,006
1899	31,819	1,105	29,037	5,196	1,548	9,990	55	2,305	35	1,648	80,962
1900	33,663	1,743	19,168	9,248	1,858	9,720	942	561	84	1,799	79,938
1901	29,992	2,005	22,707	4,435	2,530	10,313	88	1,473	(e)56	1,800	75,399
1902	29,527	978	25,593	5,023	4,648	9,210	97	580	63	1,895	77,517
1903	29,590	670	24,492	3,720	3,448	7,920	114	1,952	25	2.053	73,984
1904	28,620	411	26,478	3,784	3,800	9,765	114	1,952	55	2,045	77,024
1905	(c)	491	(c)	4,921	(c)	(c)	(c)	(c)	(c)	1,933	....

(a) Not reported in the government statistics. (b) Crystalline graphite. (c) Statistics not yet published. (d) The figures for 1897 and 1899 are exports; the enormous production in 1898 as reported in official government publications is not reflected in the exports for that year, which amounted to 24,349 metric tons. (e) The production of crude graphite in 1901 was 1727 tons.

Graphite Mining in Foreign Countries.

Australia.—Deposits of graphite were discovered some years ago near the head of the Donnelly river, in Western Australia. A syndicate, formed at the Vasse, took up and prospected several blocks, but the price of graphite at the time was so low that work was abandoned before anything certain had been accomplished. Other deposits were also found later in the same vicinity. After being allowed to lie for some years the matter has again been taken up, and regular operations and a plant are projected by the Western Australia Graphite and Plumbago Company. Five tons have been shipped as a sample to the United States, and refining works, it is said, are to be built at the mine.

Graphite is known to occur at Homebush (near Mackay), Killarney, Stanthorpe, and other localities in Queensland, but so far only at Mount Bopple has it proved of commercial value. A graphite mine is worked on the western side of Mount Bopple, 2½ miles southwest from the Netherby railway station on the Maryborough-Gympie line. The outcrop has been exposed, and a shaft sunk on it to a depth of 45 ft. An average of the graphite seams, which total 11 ft., yielded 28 per cent. of graphite carbon

and 70 per cent. of ash. The available quantity of graphite having the above composition is unquestionably very great, and the regularity of its occurrence in regular seams alternating with dark-blue coal-measure shales indicates its probable extent beyond the limit of the workings put down to exploit them.

Another mine at Mount Bopple shows graphite associated with anthracite. The anthracite, from its high percentage of carbon, was supposed to be a graphite of high quality, but it is useless for the purposes graphite is applied to, as the carbon is not in a graphitic form. At the present time parcels of graphite are being sent away for foundry purposes, for which it is said to be well suited, and the demand is increasing.

Canada.—Ontario.—A new shaft has been recently sunk at the Black Diamond graphite mine by R. McConnell, of Ottawa, who leased the property from the Ontario Graphite Company, Ltd. Attempts had been previously made to pump out and reopen the old shaft, which had caved in; but it was found impracticable to make any further use of it. The new shaft follows the dip of the vein, this having an inclination of 30 deg., and is 170 ft. in depth. The shaft is fitted with skip-track, and ore is hoisted by skip, driven by duplex cylinder single-drum hoist. The new power plant on the Madawaska river has been completed. A new conical stamp is being employed with success. The mill employs 30 men. The McConnell graphite mine, in Lanark county, seven miles north of Perth, has recently been opened.

The Anglo-Canadian Graphite Company, of Buckingham, in the Ottawa district, has assigned, with large liabilities.

Quebec.—The Calumet Graphite Mining and Milling Company, which has been operating for some time a graphite property near Calumet station on the Canadian Pacific, has decided to erect a milling plant. The property has been opened up to a considerable extent. One shaft on the slope of a hill has a depth of 80 ft., following several veins of columnar and scaly structure. A tunnel commenced at the foot of a hill will be driven toward the shaft, the bottom of which is 90 ft. above the tunnel level. There are a number of openings all over the crest of the hill, showing more or less the occurrence of the scaly variety in a disseminated form through limestone and quartz. Some 60 tons of the vein graphite mined in the shaft have been sent to the Globe Refining Company, New York, and yielded 32 tons of desirable crucible material, which has been used to good advantage in the making of crucibles by English and German manufacturers.

A bulletin (No. 877) by R. W. Ells, published by the Geological Survey of Canada, under the title "Mineral Resources of Canada," describes in detail all the occurrences of graphite in the Dominion.

The production of graphite in Canada in 1905 was 541 short tons, valued at $17,032, an increase of 100 tons over the previous year.

Natal.—The Natal Graphite Mines, Ltd., has been formed in London to work a deposit of graphite found in the Impetyini forest, 20 miles southwest of Harding, close to the East Griqualand border. It is proposed to export the graphite in a practically marketable form, ready for production as lead pencils, retorts, crucibles, etc., in the home markets.

Siberia.—Deposits of graphite are numerous. The best known are the Alibert in Irkutsk, and Turukhansk in Yeniseisk. These graphites are not excelled by those of any other part of the world, but they are not now being worked.

GYPSUM.

The production of gypsum in the United States is about 1,000,000 short tons per annum, the average value of crude gypsum being \$1.10 to \$1.20 per ton. In 1903 the total production was 1,041,704 tons, and in 1904 it was 940,917 tons, these being the statistics of the U. S. Geological Survey. The production in 1903 was the largest on record, the increase being due to a large extent to the construction of the buildings for the St. Louis Exposition, in connection with which a large quantity of staff was consumed; the decrease in 1904 was largely attributable to the cessation of this temporary demand.

STATISTICS OF GYPSUM IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production.		Imports.					
			Crude.		Ground or Calcined.		Plaster of Paris.	Total Value.
	Quantity. (b)	Value. (c)	Quantity.	Value.	Quantity.	Value.	Value.	
1896	195,553	\$583,136	180,269	\$193,544	3,292	\$21,982	\$11,722	227,248
1897	268,187	889,177	163,201	178,686	2,664	17,028	16,715	212,429
1898	281,130	864,415	166,066	181,364	2,973	18,501	40,979	240,844
1899	376,840	1,155,581	196,579	220,603	3,265	19,250	58,073	297,926
1900	432,323	1,316,255	209,881	229,878	3,109	19,179	66,473	315,530
1901	588,981	1,577,493	235,204	238,440	3,106	19,627	68,603	326,670
1902	(a) 728,998	2,089,341	305,367	284,942	3,647	23,225	52,533	360,700
1903	(a) 930,093	(c) 3,792,943	265,958	301,379	3,526	22,784	54,434	378,597
1904	(a) 840.104	(c) 2,784,325	294,238	321,306	3,278	11,276	23,819	356,401
1905	(d) 921,107	(d) 3,105,000	356,457	402,378	3,471	20,883	22,959	446,220

(a) Statistics of the U. S. Geological Survey. (b) Represents the amount of crude gypsum quarried. (c) Represents the value of the marketed gypsum including its various finished forms; the value for each of the previous years is that of the crude material. (d) Partly estimated.

The statistics for 1904 showed that normally a little more than 40 per cent. of the production is consumed for the manufacture of wall plaster; and about 30 per cent. for the manufacture of plaster of paris. About 7 per cent. is used as land plaster (fertilizer) and the remainder as crude mineral. The use of gypsum for land plaster is not as highly regarded by farmers at the present time as it was 20 years ago. The gypsum sold crude is ground into fertilizer, sold to portland cement mills, shipped to some plate glass factories where it is calcined, and sold to local mills for calcining. A considerable portion of the plaster of paris is used by plate glass factories for holding the plate of glass on the polishing tables, and is shipped to local mills in various parts of the country, where it is mixed with dried sand and retarder and sold as patent wall plaster ready to be mixed

with water and applied to the walls. An objection to the use of hard wall plaster, made in this way, has been its high sound conductivity. The introduction of fine wood fiber into the plaster, forming wood fiber plaster, has overcome this difficulty to a considerable extent.

The States producing the largest quantities of gypsum are Michigan, Kansas and Texas, New York and Iowa, which rank in the order mentioned.

The gypsum of Michigan occurs in large ledges of high purity. The most important deposits occur near Grand Rapids and at Alabaster, the latter on Saginaw bay. A third area, at present undeveloped, occurs near St. Ignace in the Upper Peninsula. The principal ledge near Grand Rapids is nearly 12 ft. thick, covered by 12 to 15 ft. of shale. The deposits are worked both by quarrying and by mining through adit levels and shafts. At Alabaster the face of gypsum is 16 to 22 ft. in hight; it is covered by 5 to 16 ft. of tough boulder clay, which is removed by steam shovels.

The gypsum deposits of Kansas form a belt trending southwesterly across the State. There are three developed areas, namely the Northern (or Blue Rapids) area, the Central (or Gypsum City) area, and the Southern (or Medicine Lodge) area. The gypsum occurs in two forms, viz., rock and earth. The rock is quarried, especially in the Northern and Southern areas. In the Northern area it is about 9 ft. in thickness; in the Southern, from 3 to 40 ft. The gypsum earth deposits are found especially in the Central area. They were of limited extent, and in several places have been exhausted. The rock deposits of the Medicine Lodge valley are worked only on a small scale, because of lack of railroad transportation. The production of Kansas in 1905 included 37,890 tons of gypsum plaster and 12,750 tons of crude gypsum.

In Texas, gypsum is worked only in the vicinity of Quanah. Both rock gypsum and gypsum earth are found in this district, but only the latter is used.

The principal sources of gypsum in New York are the quarries near Cayuga, which have been worked since 1828. The rock gypsum is mined by stripping off the cover at Fayetteville and Cayuga, by tunnels near Caledonia and Garbuttsville, and by shafts at Oakfield. The rock contains from 70 to 90 per cent. of calcium sulphate, and from 28 to 10 per cent. of calcium carbonate.

In Iowa, gypsum is mined chiefly in the vicinity of Fort Dodge, over an area of 60 to 75 square miles. The thickness of the rock varies from 10 to 30 ft. It is mined by open quarry, by adit levels, and by shafts. Practically all the Iowa gypsum is calcined into plaster.

Other States which produce gypsum in commercial quantity are California, Ohio, Virginia, Colorado, Wyoming, Oklahoma, Montana, South Dakota, New Mexico, Arizona, Utah, Nevada and Oregon. The statis-

tics of production in the United States and also in foreign countries are given in the following tables:

PRODUCTION OF CRUDE GYPSUM IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1902. (a)		1903. (a)		1904. (a)	
	Sh. Tons.	Value. (b)	Sh. Tons.	Value. (b)	Sh. Tons.	Value. (b)
California, Ohio and Virginia	101,545	$290,393	103,392	$467,113	101,809	$318,723
Colorado and Wyoming	16,051	73,372	33,549	133,347	35,778	135,045
Iowa, Kansas and Texas	295,769	807,355	307,102	1,087,045	(d) 319,080	1,027,792
Michigan	240,227	459,621	269,093	700,912	238,385	541,197
New York	110,364	259,170	137,886	462,383	158,892	432,358
Oklahoma	34,156	111,215	69,158	234,621	53,523	190,245
Other States	18,366	88,215	121,524	707,522	33,450	138,965
Total	816,478	$2,089,341	1,041,704	$3,792,943	940,917	$2,784,325

(a) Statistics of the U. S. Geological Survey. (b) Value includes that of prepared products; the value in 1905 is that of crude gypsum. (d) Of which, Iowa, 145,359 tons—$475,432.

PRODUCTION OF GYPSUM IN THE PRINCIPAL COUNTRIES. (a)
(In metric tons.)

Year.	Algeria. (b)	Canada.	France. (b)	Germany. (c)		Greece.	India.	United Kingdom.	United States.
				Baden.	Bavaria.				
1896	37,512	187,818	2,051,124	32,801	28,799	120	7,605	196,404	201,305
1897	36,750	217,392	2,004,339	40,702	26,153	51	8,187	184,287	272,493
1898	37,337	198,908	2,115,261	28,037	25,688	83	8,390	199,174	285,644
1899	39,950	221,862	1,807,454	29,419	29,727	81	6,546	215,974	382,891
1900	42,237	228,713	1,774,492	26,381	35,484	129	4,415	211,436	439,265
1901	44,025	266,605	2,385,633	28,183	3,581	671	(d)	204,045	598,529
1902	44,975	301,229	2,185,346	(d)	31,701	*Nil*	(d)	228,274	740,906
1903	41,550	285,380	1,998,804	(d)	30,894	94	(d)	223,426	945,285
1904	48,375	309,220	1,957,802	(d)	22,766	393	3,937	237,749	853,827
1905	(d)	395,453	(d)	(d)	(d)	(d)	(d)	220,416	939,201

(a) From official reports of the respective countries, except the statistics for the United States. (b) A part of the product is reported as plaster of paris. In converting this to crude gypsum it has been assumed that the loss by calcination s 20 per cent. (c) Prussia is a large producer of gypsum, but there are no complete statistics available. (d) Statistics not yet available.

Technology of Gypsum.

The manufacture of plaster of paris is described by C. O. Bartlett as follows:

The rock should be reduced by a 20x12 in. breaker to about 1-in. cube.

The crushed rock should then be dried in a rotary direct-heat dryer. The products of combustion should not pass through the material on account of the danger of coloring it; i. e., the dryer should be heated externally. It should have a good dust-settling chamber.

After the gypsum is thoroughly dried it should be thoroughly crushed by an ordinary pot or bowl crusher, and should then be ground to about 80-mesh. French buhr stones are mostly used, and are, no doubt, as good as other devices.

The ground gypsum is then passed into a calcining kettle, usually 8 ft. in diameter and 8 ft. high, with an upright shaft and stirrers near the bottom, driven with heavy gears above. The ground material is slowly

passed into the calcining kettle, wherein it soon begins to boil. More material is gradually added until the kettle is full. From gypsum rock that has been crushed and dried thoroughly, a batch can be calcined in about 1.5 hours, the time depending very much upon the dryness of the material and the quantity of the finished product.

The material contains enough moisture to boil for a short time; after it comes to a dead state it will boil a second time and in some cases even a third time. The more it boils the quicker it will set. For ordinary plaster work only one boiling is required, but for fine work or plaster of paris, two are necessary.

After the charge is sufficiently calcined, it should be immediately emptied and placed in hoppers or bins made of iron or brick, after which it is ready to be elevated and conveyed to storage bins, from which it is packed into barrels or sacks ready for the market. As soon as the kettle is emptied another charge should be immediately put in.

To manufacture 100 tons of plaster a day, the following machinery is required:

1. A crusher; estimated cost, $1000. 2. One direct-heat dryer, 48 in. in diameter and 27 ft. long, together with dust-room; estimated cost, $2500. 3. One pot or bowl crusher for grinding the material again after drying, fine enough to be ground with buhr stones; cost, $300. 4. Four French buhr stones for grinding; cost, $300 each. 5. Two calcining kettles; $200 each.

Besides the above, the plant will require hoppers, bins, conveyors, elevators and power. If it be desired to make wall plaster, in addition to the above machinery, one dryer for drying sand, and a hair picker for picking hair, and dry mixers for mixing the different materials are required.

IRON AND STEEL.

BY FREDERICK HOBART.

IN 1904, in reviewing the iron and steel industries, it was said: "If a curve should be drawn, representing the course of the iron and steel industry in the United States for a number of years, it would strongly resemble the wave line as traced by a hydrographer, the crest of each wave representing a period of prosperity, followed by one of depression, as the wave loses its force and recedes. In the case of the iron diagram, however, it would be found that each wave of prosperity rises to a higher point than its predecessor. This is to be expected, not only on account of the national growth in population and substantial wealth, but also because of developments in metallurgy, improving the quality and reducing the costs, and of the rapid progress made in the substitution of metal for lumber and other materials in construction. . . . Even the receding wave never reaches as low a point as its predecessor."

The rising wave which was apparent in the latter part of 1904 continued to rise during the whole of 1905. There was, it is true, a slight hesitation toward the close of the first quarter of the year, but it was only temporary. As soon as it began to be apparent that the year's crops would be abundant, all hesitation disappeared. From that time on the demand for pig iron and for finished material of all kinds increased with a rush; and the year closed with furnaces and mills overwhelmed with orders and obliged to use every effort to keep up with the contract deliveries. Construction of all kinds was active beyond all precedent; money for investment was abundant, and the consumption of iron and steel the largest ever known. The increase in furnaces, in furnace and mill capacity made in 1901-1903, which some conservative observers feared was excessive, has proved hardly sufficient.

IRON ORE.

The production of iron ore in 1905 was the greatest on record. The demand led to the extension of mining everywhere and to the reopening of old mines. Thus the Chateaugay and other mines in the northern Adirondacks; the Port Henry mines in the Champlain region of New York; the old mines of New Jersey and of the Lehigh Valley in Pennsylvania were all active. In the Hudson district several old mines have been reopened and promise a fair output during 1906. The production of iron ore in 1905 is given in the following table:

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.
(In tons of 2240 lb.)

District.	1900.	1901.	1902.	1903.	1904.	1905.
Lake Superior	19,095,393	20,589,237	27,571,121	24,099,550	21,822,839	34,353,456
Southern States	5,100,000	4,767,667	4,850,000	5,889,000	5,450,000	7,175,000
Other States	1,758,000	2,530,575	2,215,000	2,483,000	2,190,000	3,050,000
Total	25,917,393	27,887,479	34,636,121	32,471,550	29,462,839	44,578,456
Add decrease in stocks at Lake Erie docks	....	45,007	....	703,169	....	....
Add imports	897,792	966,950	1,165,470	980,440	487,613	845,651
Total	26,815,185	28,899,436	35,801,591	34,155,159	29,950,452	45,424,107
Increase in stocks at Lake Erie docks	690,000	....	1,214,591	....	....	......
Deduct exports	51,460	64,703	88,445	80,611	213,865	208,058
Total consumption	26,073,725	28,834,733	34,499,555	34,074,548	29,736,587	45,216,049

In this table no account is made of increase or decrease in stocks since 1903. The known stocks, chiefly those on Lake docks, were not unusually large at the beginning of 1905, and those on hand at the close of navigation season did not show any great change from the first of the year. Making allowance for stocks in furnace yards, the actual consumption of iron ore in 1905 was close to 43,000,000 tons, an average of 1.87 tons, of ore used per ton of pig iron made. The exports of iron ore were chiefly to Canada. The imports were largely from Cuba, but ores were also received from Canada, Newfoundland and Spain.

The shipments of iron ore from the Lake Superior region, of which a close record is kept, were, in 1905: Lake, 33,476,904; rail, 876,552; total, 34,353,456 tons. The total in 1904 was 21,822,839 tons, showing an increase of 12,530,617 tons, or 57.4 per cent. These figures do not include the Canadian ranges. The distribution of tonnage by ranges was as follows:

Range.	1904		1905		Range.	1894		1905	
	Tons.	Per Cent.	Tons.	Per Cent.		Tons.	Per Cent.	Tons.	Per Cent.
Marquette	2,843,703	13.1	4,210,522	12.3	Vermilion	1,283,513	5.8	1,677,185	4.9
Menominee	3,074,848	14.1	4,495,451	13.0	Mesabi	12,152,008	55.7	20,153,699	58.7
Gogebic	2,398,287	11.0	3,705,207	10.8	Baraboo	67,480	0.3	111,391	0.3

The list of shipping mines in 1905 contains 143 names against 135 in 1904, 142 in 1903, 133 in 1902 and 104 in 1901. The distribution among the ranges was as follows: Marquette, 21 against 20 in 1904; Menominee, 31 against 30 in 1904; Gogebic, 20 against 22 in 1904; Vermilion, 6, the same as in 1904; Mesabi, 65 against 55 in 1904.

In 1901 the Stevenson mine was preeminent on the producing column with a total of 1,652,021 tons. In 1905 it steps into third place, Mountain Iron being at the head of the list with 2,495,089 tons to its credit, while the Burt comes second with 1,860,452 tons and the Stevenson third with 1,428,614 tons. In 1904 there were only two mines which mined more

than 1,000,000 tons, while in 1905 there were seven which mined more than 1,000,000 tons each and one which mined more than 2,000,000 tons. Previous to 1905 the record for the production by a single mine in any year was held by the Fayal, which produced 1,919,172 tons in 1902. The shipments from mines owned or controlled by the United States Steel Corporation in 1905 were 18,783,221 tons, or 54.6 per cent. of the total

Pig Iron.

The figures collected by the American Iron and Steel Association give the output in 1905 as 22,992,380 long tons, the greatest ever recorded; the total exceeds by about 3,000,000 tons the combined make of Germany and Great Britain, our two chief rivals. The increase over 1904 was 6,495,347 tons, or 39.4 per cent.; over 1903—heretofore the year of greatest production—it was 4,983,128 tons, or 27.7 per cent. The production, under the old classification by fuel used, is divided as follows:

PIG IRON PRODUCTION ACCORDING TO THE FUEL USED.
(In tons of 2240 lb.)

Fuel Used.	1900.	1901.	1902.	1903.	1904.	1905.
Bituminous, chiefly coke	11,727,712	13,782,386	16,315,891	15,592,221	14,931,364	20,964,937
Anthracite and coke	1,636,366	1,668,808	1,096,040	1,911,347	1,228,140	1,674,515
Anthracite alone	40,682	43,719	19,207			
Charcoal	339,874	390,147	378,504	504,757	337,529	352,928
Charcoal and coke	44,608	23,294	11,665	927		
Totals	13,789,242	15,878,354	17,821,307	18,009,252	16,497,033	22,992,380

Coke was the chief fuel used, very little iron being made with raw bituminous coal in this country. Many of the anthracite furnaces even use some proportions of coke as fuel. Coke is practically the fuel with which 95 per cent. of our pig iron is made. The better classification, by purposes for which the iron is intended, divides the output as follows:

PIG IRON PRODUCTION.
(In tons af 2240 lb.)

Kind of Iron.	1902.		1903.		1904.		1905.	
	Tons.	%	*Tons.*	%	*Tons.*	%	*Tons.*	%
Foundry and forge	5,176,568	29.1	5,281,200	29.3	4,358,295	26.4	5,837,174	25.4
Bessemer pig	10,393,168	58.3	9,989,908	55.5	9,098,659	55.2	12,407,116	54.0
Basic pig	2,038,590	11.4	2,040,726	11.3	2,483,104	15.1	4,105,179	17.9
Charcoal	....	....	504,757	2.8	337,529	2.0	352,928	1.5
Spiegel and ferro	212,981	1.2	192,661	1.1	219,446	1.3	289,983	1.2
Totals	17,821,307	100.0	18,009,252	100.0	16,497,033	100.0	22,992,380	100.0

The largest proportional increase was in basic pig, and indicates the rapid growth in the use of the basic open-hearth furnace in this country.

The production by districts for three years past is given in the table following:

	1903.	1904.	1905.
New England, N. Y. & New Jersey..	782,350	880,074	1,525,094
Pennsylvania....................	8,211,500	7,644,321	10,579,127
Ohio, Illinois, Mich., Wisconsin and Minn........................	5,508,034	5,077,549	7,260,712
Maryland........................	324,570	293,441	332,096
Southern States..................	2,912,509	2,449,872	2,887,577
West of the Mississippi...........	270,289	151,776	407,774
Total......................	18,009,252	16,497,033	22,992,380

In this grouping Maryland has been put by itself because its production is chiefly from a single group of furnaces, in which imported ores are used. The great increase last year was fairly uniform in Pennsylvania and the Central West, which furnish about three-quarters of the pig-iron output. In the East the gain was above the general average, this being due to the new furnaces of the Lackawanna Steel Company at Buffalo, which came into full operation in 1905. The South made a poorer showing than had been expected, the gain over 1904 being only 17.9 per cent., while the total was 24,932 tons—or 0.9 per cent.—below that of 1903.

The approximate consumption of pig iron in 1905 was as follows:

Production, as above..............................	22,992,380
Imports..	212,465
Total..	23,204,845
Exports..	49,221
Approximate consumption..........................	23,155,624

In these figures no account is taken of stocks on hand, which did not vary greatly during the year. The approximate consumption showed an average of 463 lb. iron per capita for the country.

The production by States for three years has been as follows, in long tons:

States.	1903.	1904.	1905.
Massachusetts..............	3,265	3,149	15,987
Connecticut................	14,501	8,922	
New York..................	552,917	605,709	1,198,068
New Jersey.................	211,667	262,294	311,039
Pennsylvania...............	8,211,500	7,644,321	10,579,127
Maryland..................	324,570	293,441	332,096
Virginia...................	544,034	310,526	510,210
Alabama..................	1,561,398	1,453,513	1,604,062
N. Car. and Georgia.........	75,602	70,156	38,699
Texas.....................	11,653	5,530	
West Virginia..............	199,013	270,945	298,179
Kentucky..................	102,441	37,106	63,735
Tennessee.................	418,368	302,096	372,692
Ohio......................	3,287,434	2,977,929	4,586,110
Illinois...................	1,692,375	1,655,991	2,034,483
Michigan..................	244,709	233,225	288,704
Wisconsin and Minn.........	283,536	210,404	351,415
Missouri, Colo., and Wash...	270,289	151,776	407,774
Total................	18,009,252	16,497,033	22,992,380

The whole number of furnaces in blast on Dec. 31, 1905, was 313, against 924 on June 30, 1905, and 261 on Dec. 31, 1904.

Counting the ore, flux and fuel consumed, the production of pig iron in 1905 required the handling of between 75,000,000 and 80,000,000 tons of material. Some of it required transportation over long distances; thus nearly all the Lake Superior ore was used in furnaces a thousand miles distant from the mines.

The proportion of furnaces active was very high, especially in the last half of the year. The prices and demand for iron made profitable the operation of many of the older furnaces on the list. On Dec. 31 there were over 89 per cent. of the available furnaces in blast, and of those idle, nearly all were necessarily so, undergoing repairs after long campaigns. A recent statement prepared by the American Iron and Steel Association gives the changes in blast-furnace capacity from June 1, 1904, to Nov. 1, 1905, as follows: Completed furnaces, June 1, 1904, 28,114,000 tons capacity; new furnaces completed, 1,982,000 tons; increased capacity of furnaces rebuilt, 500,000 tons; total, 30,596,000 tons. Furnaces abandoned, 461,000 tons; furnaces idle since June 1, 1904, 1,500,000 tons; total deductions, 1,961,000 tons; furnaces standing Nov. 1, 1905, 28,635,000 tons. In addition to this, 16 furnaces which were in course of erection on Nov. 1, 1905, will have a total annual capacity of 1,830,000 tons. Of these furnaces eight stacks, with a total yearly capacity of 915,000 tons, were ready for blast in the first third of 1906.

At the middle of 1906 the total yearly productive capacity of the furnaces standing in the United States is 30,465,000 tons of pig iron. Making liberal allowance for the number necessarily under repair, or idle for unavoidable reasons, the United States is able to turn out at least 27,500,000 tons of pig iron a year. It may be added that at the close of 1905 the production was at the rate of a little over 25,000,000 tons a year.

Steel.

Steel production in the United States in 1905 showed an advance even greater than that in the pig-iron output. The increase in bessemer steel was large, but it was surpassed by that in open-hearth metal, which reached a total last year not only far beyond that of any preceding year, but also much in excess of anticipations and earlier estimates. The American Iron and Steel Association gives the following statement of production for the last two years:

PRODUCTION OF STEEL IN THE UNITED STATES.
(In tons of 2240 lb.)

Kinds.	1900.	1901.	1902.	1903.	1904.	1905.
Bessemer	6,684,770	8,713,302	9,138,363	8,577,228	7,859,140	10,941,375
Open-hearth	3,402,552	4,656,309	5,687,729	5,837,789	5,908,166	8,971,376
Crucible	131,250	103,984	121,158	112,238	92,581	121,000
Total tons	10,218,572	13,473,595	14,947,250	14,527,255	13,859,887	20,033,751
Total metric tons	10,382,069	13,689,945	15,186,406	14,756,691	14,081,645	22,157,329

The most noteworthy point about this statement is the rapid advance of the open-hearth process. In 1905 nearly 45 per cent. of the steel output was from the open-hearth furnace; and if the recent rate of progress continues, it will take only a few years for converter steel to fall into the second place, as it did in Great Britain several years ago. It is noticeable, too, that while the open-hearth process is more readily adapted to the needs of smaller works, a great part of the gain has been in the larger plants, most of which have increased the number of open-hearth furnaces, while that of bessemer converters shows only a moderate gain. The proportions of open-hearth steel made by the acid and basic processes for six years past have been as follows:

PRODUCTION OF OPEN-HEARTH STEEL.
(In tons of 2240 lb.)

Year.	Acid.		Basic.		Year.	Acid.		Basic.	
		%	Tons.	%		Tons.	%	Tons.	%
1900..........	855,529	25.2	2,547,023	74.8	1903..........	1,094,998	18.8	4,734,913	81.2
1901..........	1,037,316	22.3	3,618,993	77.7	1904..........	801,799	13.6	5 106,367	86.4
1902..........	1,191,196	20.9	4,496,533	79.1	1905..........	1,155,648	12.9	7,815,728	87.1

The increase in the proportion of basic steel, though less marked in 1905 than in the previous year, was still considerable. As all the converter steel was made by the acid process, basic metal formed last year 39 per cent. of the total steel output, against 36.8 per cent. in 1904. The basic converter has never been used to any extent in this country; and the two small plants—at Troy, N. Y., and Pottsville, Penn.—built to use that process have not been in operation for several years past.

The make of over 20,000,000 tons of steel in one year is an achievement wholly without precedent. It shows the point to which the productive capacity of the United States has been brought; it may be added, the consuming capacity, also, for all of this steel was required to meet the demand for finished forms during the year.

The various descriptions of rails made in 1905 were, in long tons:

	1904.		1905.	
	Tons.	%	Tons.	%
Bessemer steel..............	2,137,957	93.5	3,188,675	94.6
Open-hearth steel...........	145,883	6.5	183,264	5.4
Iron.......................	871	....	318	....
Total..................	2,284,711	100.0	3,372,257	100.0

Of the total output of rails, Pennsylvania furnished 1,113,841 tons, or about one-third. Twenty-four plants in 12 States rolled or re-rolled rails in 1905, as follows: New York, 1; Pennsylvania, 5; Maryland, 3; West Virginia, 1; Georgia, 1; Alabama, 3; Ohio, 4; Illinois, 2; Wisconsin, 1; Colorado, 1; Washington, 1; and California, 1. The open-hearth rails were all made in Alabama, at Ensley, with the exception of a small

quantity rolled at the Pueblo plant in Colorado. The small quantity of iron rails made was all light rails, for use in mines.

The following table gives the production of all kinds of rails in 1905 according to the weight of the rails per yard. Street rails are not included. The figures are in long tons:

Year.	Under 45 lb.	Between 45 lb. and 85 lb.	85 lb. and over	Total GrossTons.
Total for 1900.....	157,531	1,626,093	602,058	2,385,682
Total for 1901.....	155,406	2,225,411	493,822	2,874,639
Total for 1902 ...	261,887	2,040,884	645,182	2,947,933
Total for 1903.....	221,262	1,603,088	1,168,127	2,992,477
Total for 1904.....	291,883	1,320,677	672,151	2,284,711
Total for 1905....	226,580	1,599,624	1,546,033	3,372,257

The noticeable point here is the increased proportion of rails over 85 ℔. to the yard. Rails of 100 ℔. are now used by many roads, while sections of 110 ℔. and 125 ℔. have been made for special heavy service.

Changes and Consolidations.—The practical completion of the Lackawanna steel plant at Buffalo constituted the most important addition to producing works during 1905. Many improvements and enlargements of works are projected, and work on some has been begun; but generally these changes were not expected to be effective until late in 1906. The negotiations for a Southern consolidation made little progress until late in the year. In December it was announced that the syndicate headed by John W. Gates had acquired a controlling interest in the Tennessee Coal, Iron and Railroad Company. The consolidation of this company with the Southern interests of the Republic Iron and Steel Company followed, though the manner and details of the combination had not been arranged at the close of the year. The important works of La Belle Iron Company at Wheeling, W. Va., were the subject of negotiation, and the control placed in the hands of a syndicate, with a view to the ultimate disposition of the property to the best interest of the stockholders—that is, to the highest bidder. No conclusion was reached during the year.

Foreign Trade.—The exports and imports of iron and steel—including machinery—in the United States for the year 1905, as compared with the previous year, are valued as below by the Bureau of Statistics of the Department of Commerce and Labor:

	1904.	1905.	Changes.
Exports..............	$128,455,613	$142,928,513	I. $14,472,900
Imports..............	21,621,970	26,392,728	I. 4,770,758
Excess, exports.......	$106,833,643	$116,535,785	I. $ 9,702,142

The more important items of the iron and steel exports for the year were, in long tons, as given in the following table:

UNITED STATES EXPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1900.	1901.	1902.	1903.	1904.	1905.
Pig iron	286,687	81,211	27,487	20,379	49,025	49,221
Billets, blooms, etc	107,385	28,616	2,409	5,445	314,324	237,638
Bars	94,665	45,105	31,549	37,182	55,472	51,870
Rails	361,619	318,055	67,455	30,656	414,845	295,023
Sheets and plates	54,865	30,832	18,300	18,093	55,204	75,034
Structural steel	67,714	54,005	53,859	30,641	55,514	83,193
Wire	78,014	88,238	97,843	108,521	118,612	142,611
Wire-rods	10,652	8,165	24,613	22,360	20,073	6,514
Nails and spikes	43,379	29,881	35,994	42,664	45,112	47,756

Exports of pipe and fittings, not reported separately in 1904, were valued at $8,293,816 in 1905. The larger exports of rails in 1905 were 124,632 tons to South America, and 55,682 tons to Mexico.

The more important items of the iron and steel imports for the year were, in long tons:

UNITED STATES IMPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1900.	1901.	1902.	1903.	1904.	1905.
Pig iron	52,565	62,930	625,383	599,574	79,500	212,465
Billets, blooms, etc	12,709	28,164	289,318	261,570	10,807	14,637
Scrap iron and steel	34,431	20,130	109,510	82,921	13,461	23,731
Bars	19,685	20,792	28,844	43,393	20,905	37,298
Rails	1,448	1,905	63,522	95,555	37,776	17,278
Wire-rods	21,092	16,804	21,382	20,836	16,206	17,616
Tin plates	60,386	77,395	60,115	47,360	71,304	65,740

The imports of pig iron and steel billets showed considerable increases last year; but they were far below those of 1903, the totals in that year being 599,574 tons of pig iron and 261,570 tons of billets and blooms.

United States Steel Corporation.—As this corporation controls more than half the iron and steel trade of the United States, a review would not be complete without the summary of its reports, which follows:

The condensed general profit and loss account for the year 1905 is as follows: Gross sales and earnings, $585,331,736; manufacturing and producing cost and operating expenses, $440,013,432; balance, $145,318,304; sundry net manufacturing and operating gains and losses, including rentals received, $2,758,633; total net manufacturing, producing and operating income, $148,076,937; other income, including interest and dividends, $3,298,500; total income, $151,375,437; general expenses, taxes, etc., $18,570,375; balance of income, $132,805,062; interest charges, $6,710,214; balance, being the aggregate net earnings of the several companies for the year, $126,094,848; profits not yet realized in cash, $6,307,190; net earnings for the year 1905, $119,787,658.

The gross earnings above shown were $14,821,257 in excess of the banner year of 1902. The increase in gross for 1905 over 1904 was $140,926,306, and in the net, $46,611,136.

The aggregate inventories of all properties Dec. 31, 1905, equaled $113,-

387,997, in comparison with a total of $94,812,546 on Dec. 31, 1904, an increase of $18,575,451. This increase is distributed generally through the entire inventory schedule and is occasioned by the greatly increased volume of business. The undivided surplus of the corporation, which constitutes its working capital, was $69,313,794 at the close of 1905. The capital liabilities were: Common stock, $508,302,500; preferred stock, $300,281,100; funded debt, $567,893,560.

Appropriations from the net earnings given above were: Depreciation funds, $23,355,063; interest and sinking funds, $27,847,103; additions to plant and property, $26,300,000; dividends on preferred stock, 7 per cent., $25,219,677; total, $102,721,843, leaving an undivided surplus of $17,065,815 for the year.

The production of the corporation's properties for 1904 and 1905 was as follows:

	1904.	1905.
	Tons.	*Tons.*
Iron Ore Mined—		
From Marquette Range	934,512	1,359,722
From Menominee Range	1,186,104	1,871,979
From Gogebic Range	1,271,831	1,671,747
From Vermilion Range	1,056,430	1,578,626
From Mesabi Range	6,054,210	12,004,482
Total	10,503,087	18,486,556
Coke Manufactured	8,652,293	12,242,909
Coal Mined, not including that used in making coke	1,998,000	2,204,950
Limestone Quarried	1,393,149	1,967,355
Blast Furnace Products—		
Pig iron	7,210,248	9,940,799
Spiegel	100,025	158,071
Ferro-Manganese and Silicon	59,148	73,278
Total	7,369,421	10,172,148
Steel Ingot Production—		
Bessemer Ingots	5,427,979	7,379,188
Open-Hearth Ingots	2,978,399	4,616,051
Total	8,406,378	11,995,239
Rolled and Other Finished Products for Sale—		
Steel Rails	1,242,646	1,727,055
Blooms, Billets, Slabs, Sheet and Tin Plate Bars	932,029	1,253,682
Plates	404,422	780,717
Heavy Structural Shapes	313,779	484,048
Merchant Steel, Skelp, Hoops, Bands and Cotton Ties	577,384	982,782
Tubing and Pipe	710,765	911,346
Rods	84,934	84,049
Wire and Products of Wire	1,226,610	1,283,943
Sheets—Black, Galvanized and Tin Plate	757,482	924,439
Finished Structural Work	357,488	404,732
Angle and Splice Bars and Joints	72,470	150,265
Spikes, Bolts, Nuts and Rivets	46,003	61,496
Axles	62,981	149,596
Sundry Iron and Steel Products	25,787	28,236
Total	6,792,780	9,226,386
Spelter	29,963	29,781
Copperas (Sulphate of Iron)	15,805	20,040
	Bbl.	*Bbl.*
Universal Portland Cement	539,951	1,735,343

The average number of employees in the service of all companies during the fiscal year of 1905 was as follows: Manufacturing properties, 130,614; coal and coke properties, 20,883; iron ore mining properties, 12,068;

transportation properties, 14,524; miscellaneous properties, 2069; total, 180,158; total annual salaries and wages, $128,052,955. This was an increase of 32,815 employees, and of $28,274,679 in wages paid, as compared with the previous year.

The corporation is still weak on the pig-iron side. While its works made 59.4 per cent. of the total steel in the country in 1905, they furnished only 44.2 per cent. of the pig iron; and large purchases from merchant furnaces were necessary.

The Iron and Steel Markets.

The record of the markets in 1905 is well given in the reports which follow. These cover the chief primary and secondary distributing centers of the country. Naturally, under the prevailing conditions, the course of prices was upward, especially during the later months of the year. During the last quarter, in fact, the chief producing interest was, apparently, anxious to keep prices down. The various pools in which it predominated refused to advance prices at times when such a movement was really expected, and apparently warranted by conditions. There were various reasons, financial and political, for this course; but its effect was in some respects rather deceptive. A large proportion of the business during the fourth quarter of the year was done at premiums, or advances over pool quotations. Billets, for instance, on actual transactions were not within sight of pool prices; the same can be said of structural steel, plates and bars. The only branches of the trade where premiums were not paid were rails, sheets and tin plates.

The railroads were an important factor, placing large orders, not only for rails, but also for bridges, cars—especially steel cars—and locomotives. The new railroads built amounted to about 4500 miles, but a much greater quantity of material was needed for new sidings, additional tracks, and similar improvements. The construction of electric railroads was also active, and called for a large quantity of material.

Alabama. (By L. W. Friedman.)—A strike among the union coal miners, which began in July, 1904, had its influence when the year opened, but with all that the iron prices were quite high, as compared to what they were a few years back, and the market was considered firm. For three months at the beginning of the year there was a fairly good demand, prices ranging between $11 and $12.50. Spring came on with a changed feeling. Demand was not as brisk and while furnace companies were in a position to hold their product for better prices, there was no telling how long such a condition would prevail. Four months went by and there was an accumulation of iron in more than one furnace yard. But the lane was not long and the turning came with a rush. The buying was in large lots, and before the end of the summer it was officially announced that every ton of the

year's make was sold, inquiries on hand indicating the need of great quantities of iron. The fall started in with a rush for iron and prices advancing; the year closed with every ton of accumulated iron sold and the daily make, much increased, used steadily to assist in filling the orders. Quotations advanced to a basis of $14.50 per ton for No. 2 foundry iron, and almost every company in this State booked orders covering the first quarter of 1906.

The year witnessed the completion of No. 6 furnace at Ensley (Tennessee Coal, Iron and Railroad Company), the first furnace which was able to produce over 400 tons of iron a day in this section. It saw the start on the erection of a new furnace at Gadsden (Alabama Consolidated Coal and Iron Company); saw the new furnace of the Woodward Iron Company, with a daily capacity of 300 tons, started up; the re-building of a furnace at Ensley and the repairing of several other furnaces, one in particular, Alice furnace in Birmingham, property of the Tennessee Company, which was thought to be dead some time ago. The strike of the union miners caused a shortage in coal and coke, the latter in particular, but the furnace companies sought their needs in other districts and kept up the iron production here. There has been steady operation at all the iron-using industries in this State, the cast-iron pipe makers, the foundries and machine shops, the car-wheel makers, the rolling mills, and other concerns having many orders to fill and their daily needs being enormous. The district reports the greatest production of cast-iron pipe on record.

The steel industry showed in 1905 the highest production ever attained at the Ensley plant (Tennessee Company). Improvements were made at this plant. The formation of the Southern Steel Company, capital stock, $16,000,000, was announced in December; it is a merger of the Alabama Steel and Wire Company and the Underwood Coal Company. The plants of the concerns consolidated were all in steady operation, the output at the steel rod, wire and nail mills at Ensley being greater than ever before.

The Republic Iron and Steel Company kept its Gate City rolling mills in steady operation through almost the entire year. The big Birmingham mills resumed two months since. The Tennessee Coal, Iron and Railroad Company, declining to recognize the Amalgamated Association of Iron and Steel Workers, secured other help, and for the most of the year kept the Bessemer rolling mills in steady operation, working on steel. The rolling mills at Sheffield, Anniston and elsewhere in this State have done well. Some new and important industries were established.

Chicago. (By E. Morrison.)—In comparison with the previous year, 1905 was profitable and satisfactory. The first half of the year was dull and disappointing, after the revival of the iron business generally in the last quarter of 1904. But in the latter part of July came the turn of the tide,

with a sharp revival of prices and heavy sales that continued well to the end of the year.

In January pig iron sold at $17@17.50 for Northern No. 2, and $13.50 @14 for Southern No. 2, Birmingham, or $17.15@17.65 Chicago. Little iron was sold in the first month of the year, both melters and furnacemen being reluctant to enter into the usual contracts for deliveries six to eight months from the date of the order. The period of small but general sales continued to May, when the market became very weak for pig iron, with no marked strength in any branch of the iron industry. From May to July the market declined steadily for pig iron. The January prices continued fairly maintained up to May, but between this date and the middle of July Northern No. 2 went down to $15.50, its minimum for the year, and Southern sank to $11 Birmingham, or $14.65 Chicago. In June there was hardly any buying.

When the revival came, in the latter part of July, it caused active buying throughout August. Southern advanced quickly $1 and $1.25 and Northern followed a week or two later with similar advances. There was active buying by the malleable works to cover their wants for the last half of the year; the gray iron foundries were not so quick to contract for their iron, but as soon as it became apparent that the tendency of the market was upward for an indefinite time, all classes of melters poured in their orders. The maximum business of the year came in September and October, but prices continued to rise until December, when they stood $19.25 @19.75 for Northern No. 2 and $18.15@18.65 for Southern No. 2. By Dec. 15 most requirements for the first six months of 1906 were probably met, and foundrymen began to be reluctant about further contracts, in the belief that there would be a repetition of the usual January quiet and that prices would not immediately be advanced.

Highest and lowest prices for the year, with corresponding figures for 1904, are shown in the following table:

	1904		1905			1904		1905	
	Highest.	Lowest.	Highest.	Lowest.		Highest.	Lowest.	Highest	Lowest.
Lake Superior charcoal	$18.00	$14.50	$20.40	$916.50	Southern foundry, No2	$17.45	$12.65	$18.65	$14.65
Northern foundry No2	17.00	13.00	19.75	15.50	Bar iron.............	1.65c	1.25c	1.90c	1.50c
					Tank plates	1.875	1.565	1.665	1.765

Remarkably heavy buying of rails and railroad supplies characterized the market from September to the close of the year. Building materials were in very good demand for the last half of the year. Beams and channels (3-in. to 15-in.) and angles (3-in. to 6-in.) advanced from 1.665c. in January to 1.765c. in March, which latter price lasted until September, when an advance was made to 1.865c., the standard quotation for the rest

of the year. The greatest business in structural material was done in September. Iron and steel bars were strong in the last quarter. Early in the year iron bars sold for 1.65@1.70c.; by the middle of June the price had dropped to 1.50@1.55c., but from Aug. 1 there was a series of advances that brought the price for the end of the year up to 1.85@1.90c. Soft steel bars brought 1.565c. in February, but advanced in March to 1.665c., which quotation prevailed for the remainder of the year.

Cleveland. (By George H. Cushing.)—At the beginning of the year in this territory No. 2 foundry iron was selling around $16.50 in the Valleys. Before the end of the first quarter foundrymen were purchasing iron for second-half delivery. There was, during June and July, a hesitating tendency in the general market and prices eased off. Toward the middle of August, however, a strong demand for steel and buying in pig iron was resumed. From the low price of $14 for No. 2 in the Valleys the quotation mounted steadily but gradually through the remainder of the year and closed at $18 in the Valleys for No. 2, or $1.50 a ton above the opening. Most of the business done during the fall was on an average of about $16.

Toward the end of the year there was a burst of buying which suggested that period of 1902, but was on a larger and more liberal scale. In the last year and a half about 30 new furnaces have gone into blast in this territory. While the ore movement was exceptionally heavy, exceeding the receipts for any year in history, the figures show that consumption was closely abreast with the receipts. The year ended, therefore, with many of the consumers unable to supply their needs for pig iron from local furnaces and forced to rely upon outside sources for such material as they needed. The market for finished material opened strong in January and closed buoyant in December. Between times there was such active buying that the record made in 1902 was excelled. The most conspicuous feature was structural steel.

Pittsburgh. (By S. F. Luty.)—The volume of business in the iron and steel industry in 1905 exceeded all former years. The pig-iron market was particularly interesting. Production in the Pittsburgh and Valley districts was greater than ever before, and profitable prices were maintained throughout the year. This was due partly to purchases of outside iron by the United States Steel Corporation at good prices, which resulted in stimulating the market. There was a weakness in July and August, when prices of bessemer iron fell below $15 at furnace. During the year the Steel Corporation bought in these districts from 350,000 to 400,000 tons. Its first purchase was 25,000 tons in January at $16.35, Pittsburgh, which was 50c. less than the market, and the last for delivery in 1905 was 45,000 tons in October at $17.35, Pittsburgh. No purchases were made in the summer months, and prices declined. The report of the Bessemer Pig Iron Association in January showed that 86 per cent. of the furnaces using Lake

Superior ore were in operation and 96 per cent. were operating in December.

One of the strange features of the bessemer pig-iron market was that the tonnage sold was less in the months when prices were at the lowest point than at any other period of the year. A record of the large sales of the year was kept, but does not include the many sales of lots of 1000 tons or less. It shows the following tonnages; January, 111,200; February, 112,900; March, 152,400; April, 49,000; May, 36,150; June, 34,100; July, 31,100; August, 29,400; September, 153,200; October, 120,100; November, 68,150. The estimated sales for December, and from 300,000 to 500,000 tons in small lots, bring the total sales recorded here up to over 1,500,000 tons. This does not represent the production in the Pittsburgh and Valley districts, as it does not include the pig iron that went directly into steel making.

Average Prices at Pittsburgh, 1905.	Pig Iron.			Ferro-manganese.	Steel.					Nails.	
	Bessemer.	No.2 Foundry.	Gray Forge.		Bessemer Billets.	Rails.	Sheets No. 28	Tank Plate.	Steel Bars.	Wire per Keg.	Cut per Keg.
January	$16.85	$17.35	$16.35	$45.00	23.00	28.00	2.30	1.50	1.40	$1.75	$1.75
February	16.35	17.10	16.10	46.00	24.00	28.00	2.30	1.60	1.50	1.80	1.80
March	16.35	17.00	16.00	47.00	24.00	28.00	2.40	1.60	1.50	1.80	1.80
April	16.35	16.85	15.85	51.00	24.00	28.00	2.40	1.60	1.50	1.80	1.80
May	16.10	16.60	15.60	50.00	23.00	28.00	2.30	1.60	1.50	1.80	1.80
June	15.60	15.60	14.85	50.00	22.00	28.00	2.30	1.60	1.50	1.75	1.80
July	14.85	15.35	14.50	50.00	23.00	28.00	2.30	1.60	1.50	1.70	1.80
August	15.35	15.10	14.35	50.00	24.00	28.00	2.30	1.60	1.50	1.70	1.60
September	16.10	15.60	14.75	54.00	25.00	28.00	2.30	1.60	1.50	1.75	1.60
October	16.85	16.85	15.85	62.00	26.00	28.00	2.25	1.60	1.50	1.80	1.65
November	18.10	17.85	16.85	100.00	26.00	28.00	2.30	1.60	1.50	1.80	1.65
December	18.35	18.35	17.10	125.00	26.00	28.00	2.30	1.60	1.50	1.80	1.70

There was no shading of prices in any of the finished steel products, except possibly sheets and tin plate, and this was due to the fact that the capacity of the mills of the country is greatly in excess of the demand. The pool price of steel billets remained at $21 throughout the year, but no sales were made at less than $23, except in June, when only $1 premium was asked. During the last quarter billets were quoted nominally at $26. Sheet and tin-plate bars commanded a rate of from $1 to $2 above billets. In February plates, bars and structural material were advanced $2 a ton and wire and merchant-pipe prices were increased $1 a ton. The leading interest advanced sheets $2 a ton in March, but the new rates were not generally maintained. Meetings of the plate, billet and beam pools were held in July and prices were re-affirmed. Some members wanted to order an advance, but it was thought best to keep the old prices and quote premiums. Late in August structural material was advanced $2 a ton, making the prices higher than at any time since 1899 and the first quarter of 1900. Beams were advanced to 1.70c. The price of beams at the opening of the year was 1.50c., and the price was advanced to 1.60c. on Feb. 16. Narrow plates were advanced $2 a ton in September by discontinuing the differential of $2 on plates 14-in. and under, and making 1.60c. the uniform price for ¼-in. and heavier, 6¼-in. to 100-in. wide inclusive. On Oct. 1

the American Sheet and Tin Plate Company readjusted prices of tin plate. A rate of \$3.55 a box was announced on Dec. 22, 1904, but it was not maintained, sales being made as low as \$3.25 a box. The price was fixed at \$3.30@3.35 a box for 100 ℔. coke plates. Prices of ferro-manganese were more active than any other commodity. The troubles in Russia caused a famine in manganese ore, and prices began to go up. Until the last quarter the average price was around \$50 a ton for 80 per cent. ferro. In October sales were made at \$62 and in November \$100 a ton was paid. Some sales were made at \$125 a ton in December. The Republic Iron and Steel Company in November announced 2c. as the price for iron bars, the highest rate quoted for many years, and \$10 a ton above the pool price for steel bars.

One of the large deals of the year was the contract made in March by the Pittsburgh Steel Company for 192,000 tons of steel billets. It called for deliveries to begin on July 1 and to continue for one year, the Republic Iron and Steel Company to furnish 6000 tons monthly and the Carnegie Steel Company 10,000 tons monthly. The price was not made public, but it was said to be \$23 guaranteed against a decline. The contract made by the Pittsburgh Steel Company in 1904 called for 10,000 tons monthly from the Republic and 3000 from La Belle Iron Works. It ran for one year and expired on June 30. The additional 3000 tons provided for in the new contract went to the Seamless Tube Company, an allied interest of the Pittsburgh Steel Company, which completed a large tube plant in June. Late in June the steel market was considerably strengthened by another deal of the Pittsburgh Steel Company with the Carnegie Steel Company. It contracted to take its entire requirements of bessemer and open-hearth billets from the Carnegie interest for a period of years, exclusive of the contract now in force, and which will expire on July 1. The company consumes 200,000 tons of billets annually, and the new contract calls for 1,000,000 tons. The railroads were heavy buyers of equipment. The Pennsylvania placed orders for 37,321 steel cars and the Baltimore & Ohio for 10,000 cars. Deliveries extend through 1906. The plate mills and car works have enough business to keep them in full operation through the new year.

The year was remarkable for the absence of labor troubles in the iron and steel plants. Wages were adjusted satisfactorily and in some instances on a higher basis than the previous year. The sheet and tin-plate workers, owing to the conditions in those lines, made some concessions. Early in January the Pittsburgh Erectors' Association made a settlement with the International Association of Bridge and Structural Iron Workers and also with the unions of other crafts employed, as all its members had important contracts and were kept busy through the year. The strike against the American Bridge Company, declared by the Structural Iron

Workers, interfered with the completion of some contracts in this district, but the strikers did not suffer, as they were at once employed by independent concerns that had urgent contracts. The rate for puddling at the opening of the year was \$4.90 a ton, and in the last half it was \$5.50 a ton with a corresponding increase for the finishers. The puddling rate was raised to \$5.12½ on March 1, and owing to an advance in the price of bar iron, on which wages are based, the rate was increased to \$5.37½ a ton on May 1. The scale expired on June 30, and the Republic Iron and Steel Company entered into a new agreement with the Amalgamated Association of Iron, Steel and Tin Workers which provided for a base of \$5 a ton. All the independent interests accepted the new scale, and under it a rate of \$5.50 a ton for puddling went into effect on July 1.

Steel rail orders did not come in as satisfactorily as in former years, not more than 750,000 tons having been placed up to Feb. 1. The outlook was not favorable, but later orders began to come in, and the tonnage was greater than the previous year. The price of steel rails for 1906 was fixed at \$28 a ton early in September and orders were placed rapidly by the railroads. Before the end of the year over 2,000,000 tons had been contracted for. The \$28 rate has been in force since April, 1901.

Lake Superior Iron Mines.

By Dwight E. Woodbridge.

The year 1905 has been one to test the facilities of every branch of the industry; mines on all ranges; railroads to Lake Superior ports; shipping on the lakes, receiving docks at lower lake ports and railways from Erie to interior furnaces. The enormous shipment of 34,176,000 tons, so far beyond anything that had been done in past years, and so much ahead of expectations, gave a severe proof of the ability of every link in the chain. The test was exceedingly satisfactory. But all facilities are being materially extended for the traffic of 1906.

The shipments in the following years, at decennial intervals, have been: 1855, 3000 tons; 1865, 194,000; 1875, 881,000; 1885, 2,467,000; 1895, 10,430,000; 1905, 34,176,000 tons. For the decennial periods ending with each of these years the totals have been as follows, in round numbers: 1855, 78,083 tons; 1865, 864,186; 1875, 6,822,806; 1885, 17,433,226; 1895, 70,063,845; 1905, 202,188,872 tons. Out of a grand total of 300,000,000 tons of iron ore shipped from the Lake Superior region in the past 50 years, two-thirds has been the product of the final decade, and half of the last six years. Nothing so well illustrates the growth of the industry, nor argues so forcibly for the increase of ore reserves.

The tendency of the last few years toward concentration of iron ore properties in the hands of large consuming interests has been carried forward and emphasized by the events of 1905. Several of the more impor-

tant independent mining interests, which at the beginning of 1905 had mines producing standard bessemer ores in quantity, covering a wide variety of physical and chemical characteristics, have cleaned up their entire holdings, which have passed into the hands of various furnace concerns. These mines, which were among the most prominent sellers of ore on the public market, are now off the list altogether, their new owners intending to conserve these excellent ores for their own benefit. The price of explored but undeveloped properties on the Mesabi range and elsewhere has also risen, and ore in the ground now approximates 7 and 8c. a ton above royalty rates.

In this connection comes to mind the fact that the iron content of ores shipped from Lake Superior has declined materially during the past few years, and is still dropping. It is a corollary to the ownership of mines by the steel-making companies, which are not liable to make so great objection to lean ores if from their own mines as they would if the same grades came from properties of others. The extended use of these lean ores will tend to prolong the life of many a property on the various Lake ranges far beyond the limits that have been set, for there is an enormous tonnage of such ores that had never been figured as available, and would not be today if the mines were selling on the open market. Today there is far more mixing of ores at the mines to get a grade that will pass than in the old days, when the furnaceman bought grades and made his own metallurgical mixture.

All these things have tended to increase royalty rates, and while many leases are still made at the old figures of from 20 to 30c. a ton, there is a stiffening of other conditions. Ores of high grade and easily mined now bring a better rate in the ground than ever before. The average mining man has not, in the past, discriminated sufficiently between a low royalty and a high cost of mining on the one hand, and a higher royalty and a mining cost that is at least sufficiently low to make up for the difference in royalty rate on the other. The higher royalty has usually been asked, too, for an ore worth far more in the open market than that for which the ordinary rate was asked, so that the man paying a high royalty almost invariably got not only a better ore and, salable for a higher price, but also one that could be mined from 10 to 25c. a ton less than others. A few well-located and easily mined properties, containing small tonnages of high-grade standard bessemer ores, have recently been leased at prices from two to three times the average rate, and $1 a ton has been refused. But all these are exceptional and cannot be considered either as fixing a rate or as establishing a precedent.

The activity of various mining and steel-making interests in securing located ore deposits has had its natural effect in the renewed search for mines. There was in 1905 an awakening of widespread importance in the

exploration of ore-bearing lands in the Lake Superior region. This was not alone on the favorite Mesabi range, but also in other districts. Even the Vermilion range, whose exploration record has shown a series of dismal and costly failures ever since the finds at Ely in the mid 80's, is attracting much attention. Many drills are now working there, and several shafts are going down through hard jasper and iron formation rocks. The indications are what they have always been on that range, but no man can say what may be the result. It is probably true that millions of dollars have been expended in drill and other work on the Vermilion, with no return; and in spite of the extended search carried on for many years, the original Soudan location and the Ely basin so far remain the sole source of ores in the district. Explorations are in general progress on the Menominee range, where the formation is so broad and strong as to allure many of the best men in the business. Finds have been comparatively few, but more than in any other range aside from the Mesabi, and the opportunities there are generally conceded to be the best in the Lake region today. Outlying portions of the Marquette district are also under exploration, and with some measure of success. The deep development of Gogebic mines, mentioned a year ago, is still in progress. Deep work in the Newport and other mines of the central portion of the range has shown its ores to be persistent with depth, and to make under what were supposed to be base dikes as well as beneath minor dikes at higher altitudes. A little exploration has been carried on lately in the Michipicoten region, but nothing is known as to its results. On the western Mesabi range, where the larger companies were working a year ago, they have maintained steady and elaborate operations. The Oliver Iron Mining Company, especially, has become deeply interested in that part of the range, and has been continually picking up lands and developed properties. It has now under construction a railway to the center of this new district, which will be in readiness for ore traffic during 1906. The company has an enormous tonnage in sight in this part of the range, much of it a grade of ore that must be milled to extract its excess of free sand, but that, once washed, will be of good physical and chemical character and excellent in the furnace. The district is one in which large capital is required for preliminary operations, as well as for the actual mining of its ores. Its deposits are very wet, a large share of it so deep that surface mining can be carried on in only a minority of cases, and of such a character that sales to independent consumers appear improbable. In addition to the Oliver Iron Mining Company the Great Northern Railroad has heavy investments in the district, and between these two practically all the region is parceled out. In the single township 56-24, in this part of the Mesabi, it is quite probable that 300,000,000 tons may be developed. And this part of the district is owned entirely by these two concerns, chiefly by the former.

In new districts about Lake Superior exploration has progressed with considerable activity. What is called the Deerwood region, about 100 miles west of Duluth, close to the main line of the Northern Pacific road, presents almost no surface indications. This was carefully studied with the aid of the dip-needle. Lines of magnetic attraction were found, with a strike generally northeast by east and southwest by west, usually practically parallel, and at some points of their work quite numerous. Drilling has been carried along above and near these lines and a large tonnage of low-grade ores has been shown. From 20 to 30 drills have been employed in this work all the year. But it is difficult to determine much with vertical holes in a vertical formation, such as this appears to be, and very little more is known today than was apparent a year ago. A comparatively few drill-holes show a non-bessemer ore suitable for foundry use in quantity sufficient to mine. A few have shown seams of merchantable ore, but these do not seem to be economically valuable, at least not now. Scores of holes have been sunk through ore running from 20 to 45 per cent. iron, most of it high in phosphorus, some high in manganese, and the tonnage of this rock exposed is great. Some of the ore is very high in manganese, but the combined analysis is, in most cases, so low as to make it of little worth, while the manganiferous ore is generally quite heavily charged with phosphoric acid. With so much lean ore it will be surprising if somewhere, in addition to the comparatively light tonnages of good material that have so far been shown, other economically important finds are not made. Scattered along the district are from 30 to 40 drills, all hard at work. One shaft has been sunk and a cross-cut has been driven through the ore, but with no very satisfactory result.

An interest in the newly discovered Moose Mountain range, lying north of Georgian Bay, in Canada, has passed into the hands of prominent railway men of that country. A railway is to be extended to the district, ore docks will be constructed on Georgian Bay, and the shipment of ore to American furnaces will begin in a small way the coming season. The control of these deposits, which are much greater than is generally appreciated, rests with Americans who are identified with the Republic Iron and Steel Company.

Developments have been carried on a part of the year in the Atikokan region of western Ontario, with the intention of utilizing its high-grade magnetites in furnaces now under construction on Lake Superior and for shipment to other points in Canada. Developments in other parts of western Ontario have ceased, for the time being, at least, and the general opinion seems to be that little of value has been found, despite many remarkable statements that have been made.

Work has continued on the Baraboo range during the year, and there has been slow but steady mining from its single mine. Developments

in the district have not been as satisfactory as was hoped, and there is little prospect that much more will be done in a mining way for some time. The product for the year was only 75,000 tons.

Ten years ago the largest iron ore cargo on the Great Lakes was about 4000 gross tons. Four years ago it had increased to 7400 tons, this time on one of the barges of the then new Pittsburgh Steamship Company. In 1903 the biggest load of ore was 7800 tons, on the steamer *Edenborn.* The following year the *A. B. Wolvin* broke all records with 10,250 tons, and in 1905 the new *E. H. Gary,* of the Steel Corporation, carried out of Lake Superior 12,328 tons. The record is still rising, and the largest cargoes of 1906 are likely to be above 13,000 gross tons. Eight ships are now under construction for steel-making companies, the smallest of which is to be 600 ft. long, or 32 ft. longer than anything now afloat, while the largest of these new leviathans will be 602x58x32 ft., with capacity for 13,500 net tons of cargo. The lake ship-building companies have in all no less than 36 iron-ore vessels under construction, the average cargo capacity of which will be better than 9000 gross tons. These ships will have an aggregate season capacity for 5,750,000 tons of ore, and will give ample facilities for any probable addition to the ore shipments of the year. The annual increase in cargo capacity of lake shipping is a notable one, and this year has been as marked as ever. At the docks of the Duluth & Iron Range Railroad the records show that the annual increase in average load taken from the docks has been as follows, in gross tons:

Year.	Gross tons.	Year.	Gross tons.	Year.	Gross tons.	Year.	Gross tons.
1886	1,432	1891	1,881	1896	2,285	1901	4,580
1887	1,690	1892	1,854	1897	3,185	1902	4,830
1888	1,732	1893	2,070	1898	3,618	1903	5,005
1889	1,789	1894	2,275	1899	3,942	1904	5,443
1890	1,970	1895	1,981	1900	4,187	1905	5,830

The Duluth, Missabe & Northern, whose first shipments in 1893 were on ships of an average capacity of 1800 tons, in 1905 sent out loads that averaged 6101 tons.

The shipments of iron ore from the Lake region during the season of navigation in 1905 are given on a previous page, under the head of iron ore; there should be added to the total there given 169,527 tons shipped from the Michipicoten range in Canada. The shipments by ports, in 1905, were: Duluth 8,867,559 tons; Superior, 5,118,385; Two Harbors, 7,779,687; Ashland, 3,489,443; Marquette, 2,977,828; Escanaba, 5,367, 941; Peshtigo, 20,091 tons.

Iron and Steel Production of the World.

The following tables show the iron and steel production of the world, the figures being given in metric tons:

PIG IRON PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,311,949	1,161,180	87,612	2,714,298	7,549,665	23,990	2,296,191
1901	1,300,000	765,420	248,896	2,388,823	7,785,887	25,000	2,869,306
1902	1,335,000	1,102,910	325,076	2,427,427	8,402,660	24,500	2,597,435
1903	1,355,000	1,299,211	269,665	2,827,668	10,085,634	28,250	2,486,610
1904	1,369,500	1,307,399	274,777	2,999,787	10,103,941	27,600	2,978,325
1905	1,372,300	1,310,290	475,491	3,077,000	10,987,623	31,300	2,125,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	289,788	526,868	9,003,046	14,009,870	625,000	39,599,457
1901	294,118	528,375	7,977,459	16,132,408	635,000	40,950,692
1902	330,747	524,400	8,653,976	18,003,448	615,000	44,342,579
1903	380,284	506,825	8,952,183	18,297,400	625,000	47,113,730
1904	386,000	528,525	8,699,661	16,760,986	633,000	46,069,501
1905	383,100	531,200	9,746,221	23,340,258	655,000	54,054,783

The total for 1905 shows an increase of 7,985,282 tons, or 17.3 per cent., over 1904. The production of the United States in 1905 was 43.2 per cent. of the total. The three leading producers—the United States, Germany and Great Britain—made in 1905 a total of 81.6 per cent. of the world's pig iron.

The world's production of steel is given in the following table, also in metric tons:

STEEL PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,145,654	655,199	23,954	1,565,164	6,645,869	115,887	2,217,752
1901	1,142,500	526,670	26,501	1,425,351	6,394,222	121,300	2,230,000
1902	1,143,900	776,875	184,950	1,635,300	7,780,682	119,500	2,183,400
1903	1,146,000	981,740	181,514	1,854,620	8,801,515	116,000	2,410,938
1904	1,195,000	1,069,880	151,165	2,080,354	8,930,291	113,800	2,811,948
1905	1,188,000	1,023,500	403,449	2,110,000	10,066,553	117,300	1,650,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	144,355	300,536	5,130,800	10,382,069	400,000	28,727,239
1901	122,954	269,897	5,096,301	13,689,173	405,000	31,449,869
1902	163,564	283,500	5,102,420	15,186,406	412,000	34,972,497
1903	199,642	317,107	5,114,647	14,756,691	418,000	36,298,414
1904	193,759	333,522	5,107,309	13,746,051	415,000	36,148,079
1905	237,864	340,000	5,983,691	20,354,291	426,000	43,900,648

The total increase in 1905 over 1904 was 7,652,569 metric tons, or 21.7 per cent., the largest ever recorded in a single year. It was due chiefly to the great gain in the output of the United States. The three leading producers together made 83.1 per cent. of the total. The United States alone furnished 46.5 per cent. of the total. The ratio of steel to pig iron production was 91.6 in Germany, 87.1 in the United States, 61.4 in Great Britain, and 81.2 for the world's total.

Belgium.—A semi-official summary of the iron and steel production of Belgium for the full year is as follows, in metric tons:

	1904.	1905.	Changes.		1904.	1905.	Changes.
Foundry iron...	99,350	99,740	I. 390	Total pig iron....	1,287,600	1,310,290	I. 22,690
Forge iron.....	224,410	205,570	D. 18,840	Wrought iron.....	355,190	380,360	I. 25,170
Steel pig.......	963,840	1,004,980	I. 41,140	Steel ingots......	1,090,770	1,023,560	D. 67,210

The total output of iron and steel in finished forms is reported at 1,173,-020 tons in 1904, and 1,192,530 tons in 1905, an increase of 19,510 tons last year. The number of blast furnaces at the close of the year was 41, of which 35 were active and 6 out of blast. The foreign trade of Belgium for two years was, in metric tons:

	Imports.		Exports.	
	1904.	1905.	1904.	1905.
Pig and cast iron.........	347,455	507,970	39,708	41,295
Wrought iron............	88,937	97,864	529,284	628,473
Steel....................	231,801	230,169	387,063	375,229
Totals...............	668,193	936,003	956,055	1,044,997

These figures include crude as well as finished forms of iron and steel.

Canada.—The total production of all kinds of pig iron in Canada in 1905 amounted to 468,003 gross tons, against 270,942 tons in 1904, an increase of 197,061 tons, or over 72 per cent. The production in 1905 was much the largest in the history of the Dominion and exceeded that of 1902, the year of next largest production, by 148,446 tons, or over 46 per cent. Of the total production in 1905, 432,870 tons were made with coke, 4836 tons with charcoal and coke mixed, and 30,297 tons with charcoal.

The production of basic pig iron in Canada in 1905 amounted to 172,102 tons, against 70,133 tons in 1904, and the production of bessemer pig iron to 149,203 tons, against 26,016 tons in 1904. Basic pig iron was made in 1905 by three companies owning six furnaces and bessemer pig iron by two companies owning three furnaces. The production of malleable bessemer pig iron in Canada in 1905 amounted to 3300 tons; foundry pig iron, 139,528 tons; forge pig iron, 3500 tons; and white and mottled and miscellaneous grades of pig iron, including castings made direct from the furnace, 370 tons. The limestone used as flux by the blast furnaces was 290,310 tons.

The following table gives the total production of all kinds of pig iron (including spiegeleisen and ferro-manganese) in Canada from 1894 to 1905:

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1894..........	44,791	1897..........	53,796	1900..........	86,090	1903..........	265,418
1895..........	37,829	1898..........	68,755	1901..........	244,976	1904..........	270,942
1896..........	60,030	1899..........	94,077	1902..........	319,557	1905..........	463,003

On Dec. 31, 1905, Canada had 14 completed blast furnaces, of which nine were in blast and five were idle. Of the total 10 usually use coke for fuel and four use charcoal. In addition one furnace, to use coke, was being built and three coke furnaces were partly erected on Dec. 31. The most important incident of the year was the complete reorganization and continuous operation of the great works at the Sault Ste. Marie. These works succeeded in securing large contracts for rails and other material, and are now being enlarged by the building of two additional blast furnaces. The Atikokan Iron Company began to build a large furnace at Port Arthur.

The most important iron ore producer in Canada is the Helen mine on the Michipicoten range. The following are the shipments by years, in long tons:

1900.	1901.	1902.	1903.	1904.	1905.
53,470	232,505	298,430	203,413	118,355	169,527

This makes a total of 1,075,700 tons to the end of 1905.

The Moose Mountain Iron Company has begun active development work on its property in the Hutton iron range in Ontario.

Interesting experiments in the production of iron and steel in the electric furnace were made during 1905 at the Sault Ste. Marie, under the direction of Dr. Eugene Haanel, Dominion Superintendent of Mines.

The production of all kinds of steel ingots and castings was 403,449 tons, nearly all made by the basic process. The finished material reported was 385,826 tons. Of this 318,405 tons were rolled from steel, including 178,885 tons of rails, and 67,421 tons from iron.

France.—The preliminary statement of production for 1905 gives the following figures, in metric tons:

Metric Tons.	1904.	1905.	Changes.
Pig iron made	2,999,787	3,077,000	I. 77,213
Wrought iron	554,632	635,000	I. 80,368
Steel ingots	2,080,354	2,110,000	I. 29,646
Finished steel	1,482,708	1,332,000	D. 150,708

The finished steel does not include steel sold by the makers in the form of blooms or billets. The pig iron in 1905 was classified as follows: Foundry, 670,000 tons; forge, 686,000; bessemer, 157,000; basic, 1,531,000; special pig, 33,000 tons. The forms of finished steel given were: Rails, 283,000 tons; merchant steel, 635,000; plates and sheets, 351,000; forgings, 37,000; castings, 26,000 tons. About one-third of the steel ingots was made in the open-hearth furnace, the other two-thirds being converter steel.

Germany.—The figures collected by the German Iron and Steel Union give the production of pig iron for two years as follows, in metric tons:

	1904.		1905.	
	Tons.	Per cent.	Tons.	Per cent.
Foundry iron	1,865,599	18.5	1,905,668	17.3
Forge iron	819,239	8.1	827,498	7.5
Steel pig	636,350	6.3	714,335	6.5
Bessemer pig	392,706	3.9	425,237	3.9
Thomas pig	6,390,047	63.2	7,114,885	64.8
Total	10,103,941	100.0	10,987,623	100.0

Steel pig, in the German classification, includes spiegeleisen, ferromanganese, ferrosilicon and all similar alloys. The gain in the total production was 8.7 per cent. In 1904 German pig-iron production first passed 10,000,000 tons; in 1905 it fell only a little short of 11,000,000. Germany thus maintains its lead over Great Britain, though its total was less than half that of the United States.

The Iron and Steel Union estimates the consumption of pig iron in Germany in 1905 as below; the steel and finished products being reduced to their estimated equivalents in pig iron. The figures are, in metric tons: Pig iron production, 10,987,623 tons; imports, pig iron, 198,953 tons; imports, finished material reduced to pig, 164,795 tons; total supplies, 11,351,371 tons; exports, pig iron, 498,703 tons; exports, finished material, reduced to pig, 3,799,201 tons; total exports, 4,297,904 tons; balance, consumption, 7,053,467 tons. The production of pig iron was 181.3 kg. per head of population; the consumption was 116.4 kg. per head. These averages exceed those for 1904 by 12.1 kg. and 4.2 kg., respectively. The Union also gives the following statement of the production of steel in Germany for the year 1905; the figures are in metric tons:

	Acid.		Basic.	
	Tons.	Per cent	Tons.	Per cent.
Converter ingots	424,196	4.2	6,203,706	61.6
Open-hearth ingots	165,930	1.7	3,086,590	30.7
Direct castings	65,369	0.6	120,762	1.2
Total	655,495	6.5	9,411,058	93.5

The total make of steel of all kinds was 10,066,553 metric tons. The ratio of steel to pig-iron production was 91.6, the highest ever reported. The production compares with that of the previous year as follows:

Kind.	1904.	1905.	Changes.
Acid steel	610,697	655,495	I. 44,798
Basic steel	8,319,594	9,411,058	I. 1,091,464
Total	8,930,291	10,066,553	I. 1,136,262

The gain in acid steel was 7.3 per cent.; in basic, 13.1; in the total make, 12.7 per cent. Of the total, including both acid and basic, 65.8

per cent. was made in the converter in 1905, against 66.6 per cent. in the previous year.

Exports and imports of iron and steel in Germany for the year were as follows, in metric tons:

	1904.	1905.	Changes.
Exports	2,770,276	3,349,968	I. 579,692
Imports	344,967	322,907	D. 22,060

The chief items of exports in 1905 were: Pig iron, 380.824 tons; billets, blooms, etc., 472.504; bars, 323,349; rails, 284,755; shapes, 405,042; plates and sheets, 281,351; wire, 311,673; wire nails, 59,907 tons.

Imports and exports of iron ore for the year were as follows:

	1904.	1905.	Changes.
Imports	6,061,127	6,085,196	I. 24,069
Exports	3,440,846	3,698,563	I. 257,717

Imports were largely from Spain and Sweden; exports were chiefly minette ores from Luxemburg, which went to France and Belgium. Slag and slag products are important articles of commerce. The imports were 846,738 tons in 1904, and 888,665 tons in 1905, an increase of 41,927 tons. The exports were 38,587 tons in 1904, and 28,032 tons in 1905. A considerable part of the imports was basic slag, used in manufacturing fertilizers. The imports of manganese ore were 255,760 tons in 1904, and 262,311 tons in 1905; an increase of 6551 tons. Exports were small, 5536 tons in 1904, and 4116 tons last year.

The production of iron ore in Germany, including Luxemburg, was 23,444,073 metric tons; an increase of 1,396,680 tons, or 6.3 per cent., over 1904. Adding the imports and deducting the exports, gives the approximate consumption of iron ore in 1905 at 25,830,706 tons; an average of 2.35 tons per ton of pig iron made.

Mexico.—While there was little actual change during the year, and production is still limited, several companies, one supported by English capital, one by German and the rest by money from the United States, have been making explorations, chiefly in Durango. It is highly probable that 1906 will see the establishment of two blast furnaces at least in the Republic, to be followed by works for producing finished iron and steel.

Russia.—In Volume XIII a full statement of production for the 15 years, 1890—1904 inclusive, was given; probably the most complete ever published. For 1905 the collection of statistics has been rendered difficult by the disturbed condition of the country. An expert estimate of production is given in the subjoined table, comparison being made with the official statement for 1904; the figures are in metric tons.

About 70 per cent. of the steel is made by the open-hearth process.

The heavy falling off was due to the general depression of business resulting from political conditions.

	1904.	1905.	Changes.
Iron ore	5,272,300	4,050,000	D. 1,242,300
Pig iron	2,978,325	2,125,000	D. 853,325
Steel ingots	2,811,948	1,650,000	D. 1,161,948
Steel rails	401,541	275,000	D. 126,541

Spain.—The production of iron ore in Spain in 1905 was the largest on record. The total ore mined and exported was as follows, in metric tons:

	1904.	1905.	Changes.
Ore mined	7,964,748	9,395,314	I. 1,430,566
Exports	7,291,941	8,590,482	I. 1,298,541

The exports in 1905 were: To Great Britain, 5,845,895 tons; Germany, 1,946,799; Belgium, 314,203; France, 251,716; United States, 213,203; other countries, 18,666 tons.

The production of iron and steel in Spain for two years was as follows, in metric tons:

Kind.	1904.	1905.	Changes.
Pig iron	386,000	383,100	D. 2,900
Wrought iron	53,177	52,250	D. 927
Bessemer steel	93,100	113,664	I. 20,564
Open-hearth steel	100,659	124,200	I. 23,541
Total steel	193,759	237,864	I. 44,105

This shows a considerable gain in steel made. The most important manufacturing concern in the country is the Sociedad de Altos Hornos de Vizcaya, which is credited in 1905 with 209,000 tons of pig iron, 55,000 tons of open-hearth steel and all the bessemer steel made.

United Kingdom.—The figures collected by the British Iron Trade Association show in 1905 the largest pig iron production ever reported; the total being 287,418 tons more than in 1899, heretofore the year of greatest production. The output in long tons for two years compares as follows:

	1904.	1905.	Changes.
Forge and foundry	3,841,975	4,276,943	I. 434,968
Bessemer	3,362.883	4,070,222	I. 707,339
Basic	1,192,120	1,057,999	D. 134,121
Spiegel and ferro	165,680	187,573	I. 21,893
Total	8,562,658	9,592,737	I. 1,030,079

The increase in the total make was 12 per cent.; certainly a notable gain. Not all the iron made was used, however, for an expert estimate of stocks in public stores and makers' hands at the close of 1905 is 900,000 tons; an

increase of 500,000 tons over the previous year. As with pig iron, the steel production showed a large increase in 1905. For a number of years past the make of open-hearth steel in Great Britain has largely exceeded that of converter steel. The totals for two years past are given in the following table:

	1904.		1905.		Changes.
	Tons.	Per ct.	Tons.	Per ct.	Tons.
Open-hearth	3,245,346	64.6	3,879,748	65.9	I. 634,402
Bessemer	1,781,533	35.4	2,009,712	34.1	I. 228,179
Total	5,026,879	100.0	5,889,460	100.0	I. 862,581

The division of the make according to the acid and basic processes in 1905 was as follows:

	Acid.		Basic.		Total.	
	Tons.	Per ct.	Tons.	Per ct.	Tons.	Per ct.
Open-hearth	3,084,510	52.4	795,238	13.5	3,879,748	65.9
Bessemer (converter)	1,117,731	19.0	891,981	15.1	2,009,712	34.1
Totals	4,202,241	71.4	1,687,219	28.6	5,889,460	100.0

The proportions of acid and basic open-hearth steel showed little change in 1905; but there was a considerable increase in basic converter steel, as compared with 1904. British practice leans toward the acid process, and will doubtless continue to do so as long as supplies of low-phosphorus ores are found in England and Scotland, and in the Spanish mines, from which so large a quantity of ore is used in the British furnaces. The basic process, naturally, is most in use in the important Middlesborough district, which uses basic ores.

The total number of converters reported in 1905 was 76, of which 60 were in use and 16 idle during the year. The average make per converter in use was 33,495 tons. The average number of open-hearth furnaces in operation during 1905 was 383, giving an average output of 10,130 tons per furnace. At the end of the year there were 514 open-hearth furnaces in Great Britain, of which 376 were acid and 138 basic furnaces. There were also 15 new furnaces under construction.

A partial report of the make of finished steel is as follows:

Kind.	Bessemer.	Open-hearth.
Blooms and billets	286,082	402,535
Rails	951,552	108,953
Bars, including tin-plate bars	288,980	839,415
General merchant steel	187,973	....
Plates and angles	....	1,765,111
Totals	1,714,587	3,116,014

This statement omits some important items.

The British foreign trade is of great importance, and is more extended than that of any other nation. Its total value, including machinery, is estimated by the Board of Trade returns as follows, for two years:

	1904.	1905.	Changes.
Exports..	£53,587,013	£60,524,755	I. £6,937,742
Imports.	12,529,212	13,128,270	I. 599,058

Exports were chiefly of finished products in various forms. In 1905 machinery and new ships built for foreign countries furnished £28,696,913 of the total value given. Imports are largely of crude or semi-finished products, such as steel blooms, billets and the like.

The chief items of the iron and steel exports were as follows, in long tons:

	1904.	1905.	Changes.
Pig iron	810,934	981,891	I. 171,957
Wrought iron	170,505	183,406	I. 12,901
Sheets	385,408	407,021	I. 21,613
Plates	152,337	204,503	I. 52,166
Rails	525,371	546,644	I. 21,273
Steel ingots, etc.	122,930	151,809	I. 28,979
Tin plates	359,634	354,951	D. 4,683
All other kinds	735,723	891,290	I. 155,567

The total increase in quantities in 1905 was 458,673 tons, or 14.1 per cent. Exports of tin plates to the United States were 71,862 tons in 1904, and 63,052 tons in 1905; a decrease of 8810. Exports of pig iron to the United States increased from 56,027 tons in 1904 to 172,789 tons in 1905; a gain of 116,662 tons.

The chief items of the iron and steel imports were, in long tons:

	1904.	1905.	Changes.
Pig iron	133,734	129,039	D. 4,695
Wrought iron	104,242	105,960	I. 1,718
Steel ingots, etc	522,706	603,949	I. 81,243
Structural steel.	122,954	123,001	I. 47
All other kinds.	308,350	393,980	I. 85,630

The total increase in quantities in 1905 was 63,943 tons, or 4.9 per cent. The gain was chiefly in semi-finished material.

Imports of iron ore in 1905 were: Manganiferous, 285,583; other, 7,065,151; total, 7,350,734 tons, an increase of 1,249,978 tons over 1904. Of the total last year 5,764,443 tons were from Spain.

The Progress of the Metallurgy of Iron and Steel in 1905.

By Bradley Stoughton.

This year has seen too many important changes for all to be recorded in a brief review. It has been a banner year not only in production, but in the application of science to practice. The numerous exhaustive abstracts of literature, published in many periodicals, makes it undesirable

to attempt anything of this kind in an annual publication. The improvements and changes which seem to have permanent value are as follows:

The Blast Furnace.

Ores.—No new ranges, or deposits of first importance, were discovered in 1905. The history of the Lake Superior district repeated itself, however, in that the supply in sight at the end of the year was as large as at the beginning, in spite of a very big output; in other words, the more work that is done the larger is the amount of ore known to exist. The oldest of the Lake Superior ranges was opened in 1854; the greatest producer, the Mesabi, in 1892; the estimated ore in sight at the end of the last century was 750,000,000 tons; the estimated ore in sight at the present is a little less than 1,000,000,000 tons, not including the more than 300,000,000 tons produced to date.

The usual predictions of the exhaustion of the world's iron ores at a none too distant date were almost as frequent as usual in 1905, in spite of the lessons of history. R. Anspach,[1] however, basing his assertion on an exhaustive report of the world's iron ore supply to the Swedish Parliament by Professor A. E. Törnebohn, President of the "Sveriges Geologiske Undersokning," has stated: "The supply of ores to furnish the world's iron consumption will presumably never be exhausted."

Furnace Construction.—It will be remembered that when the first 100-ft. stacks were erected a number of engineers protested against this hight as excessive, and it was prophesied that time would see the limit reduced to 80 ft. This prediction has been partly justified, and 90 ft. is not now exceeded by engineers. Moreover, the large furnaces projected during 1905, with a capacity of 600 tons per day, are to have a hight of only 85 ft., and bosh diameter of 22 ft.

Dried Air.—The great interest and discussion provoked by the announcement by James Gayley, in October, 1904, of the results of his experiments at the Isabella furnaces to increase the fuel economy and regularity of working by employing blast from which the greater part of the moisture had been removed by refrigeration,[2] continued to occupy the greatest share of attention in this subject during 1905. Among foreign writers and ironmasters, the results published by Mr. Gayley were oftentimes received with skepticism, based on theoretical grounds, and a large number of elaborate and delicate calculations have been made to prove the impossibility of the saving effected in the furnace in question having been due entirely to the drying of the blast used. Some even went so far as to doubt the ac-

[1] "Iron Resources of the World," *Engineering and Mining Journal*, October 7, 1905.
[2] "The Application of Dry-Air Blast to the Manufacture of Iron," by James Gayley. *Journal* of the Iron and Steel Institute, No. II, 1904, pp. 274-322.

curacy of the records published.[1] Those who accepted Mr. Gayley's results as accurate explained them rather on the ground that the furnace had been working poorly before the dry air was used, and the improved result was a combination of dried air and better care on the part of those in control.

In May, 1905, Mr. Gayley furnished further data by publishing the records of the Isabella furnaces for the months of November, 1904, to March, 1905, inclusive.[2] A better comparison was afforded by changing the dry blast from No. 1 furnace to No. 3. The result was an increase in the production of No. 3 furnace, and a decrease in the average fuel consumption, with the corresponding reverse change in the working of No. 1.

Meanwhile, the rationalism of the astonishing amount of saving has been exposed by two able discussions of the subject.[3] [4]

The economy of using the air dried by refrigeration may be obtained in three ways: (1) increased production; (2) increased burden; and (3) lower duty on the blowing engines, on account of the colder air being denser. The great advantage, however, is in the regularity of working of the furnace, because atmospheric air may suffer a rapid and large variation in the amount of entrained moisture which it carries into the hearth of the furnace, so that the burden must leave a safe margin of fuel to take care of a sudden loss of heat due to this cause. Since the refrigerated blast is not subject to this irregularity, no such margin is required, and, furthermore, the product is more uniform. The actual amount of heat saved in the hearth of the furnace by drying the blast is never a large fraction of the total heat supplied to it, and several metallurgists have pointed out that an equivalent amount of heat might be furnished by heating the undried blast to a higher temperature, but such an arrangement would not automatically counteract the irregularities due to variation in moisture.

The original refrigerating plant installed at the Isabella furnaces in 1904 has not been extended elsewhere during 1905. That this is not due entirely to lack of appreciation of the advantages such a process affords is evident from the opinions expressed by several ironmasters, but the large amount of capital to be invested in plant makes a responsibility that must not be assumed hastily. Moreover, it is the opinion of many persons that, now the disadvantages resulting from irregularity of hearth temperature have been so fully demonstrated, and the astonishing improvement caused by even a partial accomplishment of uniformity has

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[1]One is reminded that a similar attitude was assumed by many foreigners when an apparent conflict of theory and practice existed over the results obtained by what is now called "high-speed steel," and that the expression "Yankee bluff" was frequently used in this connection.

[2] *Journal* of the Iron and Steel Institute, No. I, 1905, pp. 256-320.

[3]"Notes on the Physical Action of the Blast Furnace," by J. E. Johnson, Jr., *Trans.* of the American Institute of Mining Engineers, Vol. XXXVI, 1906. (Advance proof.)

[4]"Iron, Steel and Other Alloys," second edition, 1906, by Henry M. Howe.

been proved, some cheaper or better means of attaining an equal result may be found.[1]

Stoves.—A method of cleaning the dust from the checkerwork of hot blast stoves has been devised by Richards and Lewis, and adopted by the Glengarnock Iron and Steel Works in England.[2] The checkerwork is divided into six vertical compartments, any one or more of which may be closed by a valve. In order to remove the accumulated dust it is only necessary to allow all the blast to pass through each one, or any, of the compartments, the velocity being of course nearly six times the normal. In this way the labor and loss of time occasioned by cooling off the stove and cleaning it by hand is said to be avoided. It is also claimed that the manager is given a certain elasticity in the temperature to which the blast may at will be heated.

Gas.—The cleaning of the furnace gas has received a good deal of attention abroad, and some in this country, but no unusual advancement has been made public.

No other installments of gas engines for blast furnace gas have been made in the United States.[3] The Lackawanna Steel Company is reported to have increasing success in its electric generation from blast furnace gas engines, although a number of accidents have occurred by leakage of the gas. This is not, of course, to be charged against this principle of blast furnace gas utilization, since it is known to be an avoidable difficulty, even though met with at many works abroad at the present time.

The same general tendency existed in 1905 as formerly for an increase in the size of the gas engine units, and there seems to be no logical reason why engines of 3000 or more horse-power should not become very common, and 6000 h.p. units are now sometimes employed. In Vol. II, 1905, of *Revue de Metallurgie,* on pp. 33 to 95, is a very complete and instructive treatise on the subject of utilization of blast furnace gases, by Ch. de Mocomble.

Fixation of Atmospheric Nitrogen.—Whether or not blast furnaces and Bessemer converters will ever be blown with oxygen less diluted with nitrogen than the atmospheric air, has not been definitely answered as yet. That such a desideratum approaches nearer to being accomplished every year is a matter of history. The possibilities resulting from such an attainment are beyond our calculations and the only obstacle now remaining is the cost of producing the oxygen. Some means of employing the residual nitrogen would materially cheapen the use of oxygen separated out of the air, but nitrogen is so inert chemically that, although its compounds have good commercial value, the cost of combining, or "fixing,"

[1]The Carnegie Steel Company is now (May, 1906) installing air refrigerating apparatus at others of its blast furnaces.
[2]*Iron and Coal Trades Review*, December 30, 1904.
[3]At least two others are now (May, 1906) being installed, however, one set going in at the Carrie Furnaces, the electricity from which is to be used across the river at the Homestead works.

it has been too great. Important advances have been made in this regard during 1905. The two methods which are commercially most promising are: (1) "Sparking," or producing combination by means of an electric arc; and (2) producing a reaction between calcium carbide and atmospheric nitrogen, which forms a valuable fertilizer.[1]

That the advantages obtained by drying the air used in the blast furnace could be obtained as well by enriching undried air with oxygen proportioned automatically to the amount of moisture will not be doubted. The interest awakened by Mr. Gayley's article, together with the commercial accomplishment of the fixation of nitrogen, may well lead to important advances in this phase of the blast furnace operation.

Coke.—In almost all fields extensive building of beehive coke ovens has been in progress during 1905. There has also been an important advance in the use of by-product coke in blast furnaces and iron foundries, the prejudice against it having now almost disappeared. The number of by-product coke plants in connection with steel works has been increased, as well as the total number of ovens in the country, which is now 2380 according to the Otto-Hoffman system, and 1255 according to the Semet-Solvay system. The important advances of the year in by-product coke oven practice[2] are increased yield of coke and gas per oven, increased use of oven gas for open hearth melting, improved quality of gas, and long distance transmission of gas under pressure.

Blast Furnace Records.—Furnaces D, E, J, and K, of the Edgar Thomson plant of the Carnegie Steel Company, beat the world's record by making 77,242 tons of pig iron during March, 1905. Furnace K, on March 30, 1905, made 918 tons of pig iron, which is the largest 24-hour output ever made.[3]

Bessemer Process.

There were no new Bessemer plants built in 1905, and only one plant has been built in the last 10 years, though another will be completed in 1906. The year's production was enormously increased, however, due to rapid driving. The time of blow has been reduced by decreasing the silicon, and heats are occasionally blown with this element as low as 0.7 per cent. The production of railroad rails is still the largest item of Bessemer production, so that the manufacture of such a large tonnage of rails by the basic open hearth process in 1904 and 1905 is observed with some apprehension by the Bessemer managers. This is increased by the fact that the open hearth plants have been uniformly busy, even when there was a slight falling off in demand for Bessemer rails, such as existed in 1904, and by the fact that the special committee of the American Society for Civil Engineers reported in favor of basic open hearth steel for rails. It

[1] "Fixation of Atmospheric Nitrogen," by Philippe A. Guye; *Electrochemical and Metallurgical Industry*, Vol. I, 1906, pp. 136-139, gives recent information on this subject. See also *Harper's Monthly Magazine*, April, 1906, R. K. Duncan.

[2] See especially "Coke Making in the United States," by Edward W. Parker; *The Iron Age*, Jan. 4, 1906, pp. 912.

[3] In March, 1906, the four Duquesne blast furnaces of the Carnegie Steel Company made 78,120 gross tons of pig iron.

is reported that the United States Steel Corporation will erect a rail mill at its new Indiana plant which will roll basic open hearth steel exclusively,[1] and a new rail mill for the same purpose is now erecting at the Edgar Thomson works.

Open Hearth Process.

General Advances.—The important changes in this country in 1905 have been decreased proportion of scrap used in the charges, increased amount of ore, increased amount of "hot metal," or fluid pig iron direct from the blast furnace or from a mixer, and increased amount of oil used for fuel outside of the districts where natural gas is available. The first two changes are natural developments, foreshadowed in recent years. In the use of oil we may expect very important advances in the near future. The greatest advantage of this fuel is the cheapness with which it can be transported. Per unit of weight and per unit of volume there is more heat in oil than in any other fuel, besides which the handling and transferring of it is accomplished with the minimum of labor. The next important point in considering the use of oil is the great saving in the repairs to the furnace and regenerators, and the longer campaigns that are possible. The means by which oil may be introduced into the furnace saves greatly in the life of the roof, and, moreover, ordinary leakages in ports and regenerator walls are unimportant, when oil is the fuel. It gives a greater regularity in furnace working, on account of its uniform composition, and better temperature control, on account of ease of regulation of fuel supply. The simplicity of applying oil fuel by means of an ordinary blow-pipe, the ease of loading and unloading, by means of pumps, the absence of ashes or dirt, and of accumulations in gas mains, all render the cost of labor less with oil than producer gas. Oil is usually injected into the furnace through a blow-pipe by means of which it is atomized with steam or compressed air.

Production.—The production of open hearth steel has been trebled since the beginning of the twentieth century, having increased very much more rapidly than any other process. The principal increase is in the basic process, and during 1905 this process increased more rapidly than the acid, even for the manufacture of steel castings.

Special Pig and Ore Processes.—The Talbot process has been extended more than any other of the special open hearth processes during 1905 and there are several new furnaces of this type which were put in operation, or which will shortly be started up[2]. The successful employment of this process in fixed furnaces was also described[3]. The Morell process,

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[1] See *The Iron Age*, Feb. 8, 1906, pp. 513.

[2] See *The Iron Age*, Jan. 18, 1906, or *Iron and Steel Magazine*, Feb. 1906, p. 169, for all the present Talbot plants in operation.

[3] "The Continuous Steel Process in Fixed Furnaces," by S. Swizycki. *Journal* of the Iron and Steel Institute, Vol. I, 1905, pp. 112-121.

controlled by the Carnegie Steel Company, is much used in the works of that company.

Duplex Process.—It is interesting to note that the combined bessemer and open hearth process, inaugurated at Witkowitz, Austria, some years ago, is still in operation there.[1] The metal is first blown in acid-lined bessemer converters to rid it of almost all of the silicon, manganese and carbon, but none of the phosphorus, after which the operation is completed in a basic open hearth furnace, which takes about three hours. A large output is obtained, and the loss of metal is 10 per cent. In addition to the blown metal, lime, iron ore, pig iron and steel scrap are charged into the open hearth furnace.

The duplex process in use for over a year by the Tennessee Coal, Iron and Railroad Company employs an acid converter and an acid open hearth furnace. The converter eliminates the carbon, silicon and manganese, and pours its metal into a primary rolling acid open hearth furnace of 250 tons capacity. This primary furnace supplies liquid metal to ten 50-ton basic lined open hearth furnaces, which is said to take the place of steel scrap, the price of which is high. The details of the process are kept secret by the manufacturers, but the results are evidently satisfactory.

Another and similar process has been developed at the same plant by the former superintendent, Henry Knoth. In this process the slag resulting at the end of one operation is returned to the basic open hearth furnace, enriched by additional limestone, and materially assists in the purification of the metal added for the next operation, which preferably consists of bessemer metal blown down to about 1 per cent. of carbon. This process is in use at the Monterey plant in Mexico and is said to give excellent results,[2] saving in basic materials by increasing the production and the life of the furnace hearth, and in handling slag. The Knoth process was patented on May 2, 1905 (U. S. patent, 788,650).

Foundry Practice.

Molding.—The most important advances in foundry practice in 1905 have been in connection with molding, where progress has taken place along two lines: (1) the extension of the field for machine molding (as for example, the installation of large hand-ramming machines for floor work,[3] the extension of mechanical appliances, and the molding of car wheels),[4] and (2) the use of permanent molds,[5] made of carbon or similar material. The second of these processes is still in the experimental stage, but important advances may be expected to follow.

Pig Iron Direct from the Blast Furnace.—There has also been a notable

[1] See *Stahl und Eisen*, 1905, Vol. XXV, pp. 1125-1127.
[2] See *The Iron Age*, July 13, 1905.
[3] "Molding Machine Practice," by F. W. Hall, *The Foundry*, Sept., Nov. and Dec., 1905, Vol. XXVII.
[4] "Machine Molding and Continuous Casting of Car Wheels," *The Iron Age*, Jan. 4, 1905, pp. 1-8.
[5] "Permanent Molds and Carbon Cores," by Henry C. Caldwell, *The Foundry*, Feb., 1906, Vol. XXVII, p. 300.

increase in the amount of direct metal used. In this connection mixers have been tried, similar to those employed at the bessemer steel works. It has also been found advantageous to mix steel in with the melted iron from the blast furnace, in order to close the grain.

Steel Castings.—There has been an increase in the production of steel castings, as compared with those of gray and malleable cast iron. In this connection the use of small bessemer converters of special design, and usually of one to four tons capacity, has been the most remarkable development. No less than six of these have been erected and put in operation during the year, and eight more will be installed early in 1906. This is about double the number of open hearth furnaces built and building for steel casting work. The advantages of the small converter type of furnace are: (1) ability to make unusually hot steel for small sized castings; (2) great flexibility in analysis, whereby a customer may be supplied, at almost no extra cost, with castings of any desired analysis, including the different alloy steels; and (3) the ability to increase or decrease the output at will and with no additional cost. The installation of these converters has been made chiefly by gray iron foundries and machine shops of manufacturing concerns to supply their own needs, although some foundries have been built especially for jobbing work.

The chief extension of the steel casting industry is evidently due to the preference of engineers for a more ductile material than is obtained in gray iron castings, since malleable cast iron is also advancing proportionately greater than its gray rival. One of the more important causes of the increased use of steel is the demand for it for railroad rolling stock, where steel castings have come into very great favor.

Thermit.—The use of the thermit process in foundries, for heating the ladle of iron or steel, for heating the tops of the molds to decrease the size of the risers, and for removing defects in castings has increased greatly. The only new development has been the perfection of the process of igniting the thermit on the casting itself, instead of igniting it in a crucible and then pouring the metal on the desired point[1].

Electric Smelting.

Pig Iron.—Much attention has been devoted to this subject during 1905, founded chiefly on a report prepared with great thoroughness by a commission appointed by the Canadian Government, and under the direction of Dr. Eugene Haanel, Superintendent of Mines, Ottawa, which was published in the latter part of 1904[2]. Later, the Canadian Government appropriated $15,000 for the purpose of making a working test of the practicability of commercially producing pig iron from Canadian ores by electri-

[1] In this same connection it may be well to note that the use of the thermit for pipe welding has also increased during the year.

[2] "Report of the Commission Appointed to Investigate the Different Electro-Thermic Processes for the Smelting of Iron Ore and the Making of Steel in Operation in Europe," Department of the Interior, Ottawa, Canada.

city, in competition with coke blast furnace smelting. Cheap water power and iron ore are to be found in many localities in Canada, together with charcoal or peat, but a sufficient supply of coking coal for blast furnace smelting is not found except in Nova Scotia and near the Pacific coast. The first question was, whether the pig iron could be economically produced using charcoal or peat coke for reduction and electricity for heat. The principal ores available are magnetites often high in sulphur and titanium, and, therefore, the second question to be answered was, whether either, or both, of these classes of ores could be smelted to produce a pure material. The experimental electric furnace to answer these questions was built upon the Sault Ste. Marie river according to the designs of Dr. Paul Héroult, who also directed the tests, Dr. Haanel being present and representing the Canadian Government. The Lake Superior Power Company loaned a building and 300-h.p. generating plant free for four months. Tests were begun in earnest in the middle of February, 1906, and extended until March 5. Hematite ores were smelted at first, and then Canadian magnetites from the different sources of supply. In all 150 casts were made, giving 55 tons of pig iron[1]. On February 24, Dr. Haanel is reported to have sent the following telegram to Hon. Frank Oliver, Minister of the Interior:

"Successful demonstrations of all points stated in my memorandum on electric smelting of Canadian iron ores requiring investigation. Output greater than figure adopted by Harbord in report of Commission. Successful smelting of magnetite and desulphurization of pig iron. Successful substitution of charcoal, and, therefore, of peat, for coke. Consumption of electrodes insignificant. Production of nickel pig of fine quality from roasted pyrrhotite. Forty tons of pig have so far been produced. Process admits of immediate commercial application. Experiments will be completed in about two weeks."

Since that time Dr. Haanel has confirmed these results in an address delivered to the Canadian Club in Toronto[3]. He states that metal can be produced containing only a few thousandths of one per cent. of sulphur, and of any desired silicon, and that poor ores, costing only $1.25 per ton, are valuable from a commercial standpoint, and will produce high-grade pig iron. Roasted pyrites from sulphuric acid manufacture can be employed, and iron ores containing as much as 5 per cent. of titanium.

The cost of the production of a ton of pig iron under the conditions existing at the "Soo" is said to be $10. The Lake Superior Power Company has purchased the plant, and it is said that it will operate it continuously.

Steel.—The Halcomb Steel Company, of Syracuse, N. Y., has also installed the Héroult electric process for the production of high-grade steel

[1] See *Electrochemical and Metallurgical Industry*, Vol. IV, April, 1906, pp. 124-126.
[2] *Ibid.*, March, 1906, p. 84.
[3] *Ibid.*, May, 1906, p. 164.

from pig iron, the intention being to make chiefly tool steels in competition with the ordinary crucible process. The great advantage claimed is that very pure qualities of steel may be produced, including any desired variety of alloy steel. A start was made in early April, 1906.

The statement has been made that "from the very start the operation has been satisfactory in every respect." The experiment is too new, however, to warrant conclusions being drawn at the present time, and I am, therefore, inclined to think that the statement quoted is premature.

In England the Sheffield Steelmakers, Ltd., has acquired the rights of the Héroult process for steel, and it is expected that a plant will be installed at an early date (see the *Ironmonger*, July 22, 1905).

Future of Electric Smelting.—From the commercial standpoint the electric production of iron appears to be desirable under the circumstances of cheap, impure ores, cheap power for the production of electricity, and costly coke. Aside from this commercial aspect, which will always receive the first consideration, the advantages of electric production seem to be the purity of the product, especially from sulphur which cannot be reduced to low proportions by any other means, and perhaps also from those elusive evils oxygen, hydrogen and nitrogen, though this is yet to be proved.

Pyrometers.

The Holborn-Kurlbaum optical pyrometer, which promised to be of very great service in the measurement of temperatures at steel works, is now withdrawn from this country as a result of unfavorable patent litigation. The Morse thermo-gage is much used, especially for low heats. For high temperatures it has a severe effect on the eyes of the observer. The Wanner instrument, though not used so much at steel works, is becoming more important for laboratory use. There seems to be no reason, except lack of advertising, why it has not been used in the works as extensively. A most valuable pamphlet on this subject appeared during the year, which should be studied by every metallurgist at the earliest opportunity.

A very important advance in pyrometry is the work of the Bureau of Standards at Washington, where instruments may now be sent for calibration at very low cost. The greatest obstacle to the successful and constant measurement of all temperatures obtained in iron and steel manufacture and treatment has thus been overcome. The irregularities to which high temperature pyrometers are subject, and their untrustworthiness in the hands of any but trained experts, may now be avoided. Instruments are available of more than the necessary degree of accuracy, and so simple to use and read that they can be placed in the hands of young mechanics or intelligent men around the plant. By having a few extra

instruments, and employing a regular system of checking up at intervals by the Bureau of Standards, admirable pyrometric work may be regularly carried on in connection with every operation. The Bureau of Standards is so crowded at present that it seems to take some months for an instrument to be calibrated and returned. This means that works must have plenty of instruments so as to allow for this delay, but the price of them is so small compared with the enormous advantages of the now easy employment of them that a small hindrance of this kind will not be considered seriously.

Alloy Steels.

High Speed Tool Steels.—Although no new high speed steels were introduced during the year, an important addition was made to our knowledge of the subject by a scientific study of the effect of tungsten and molybdenum, the two alloying metals used in this connection, published by Dr. H. C. H. Carpenter in the *Journal* of the Iron and Steel Institute, No. 1, 1905, for which Dr. Carpenter was awarded the Carnegie medal, and which ably supplemented the article on the same subject by Gledhill in the preceding volume of the same journal. A great deal of study was devoted to these tool steels during the year, for the results of which the reader is referred to the regular bibliographies of the *Iron and Steel Magazine* and the *Journal* of the Iron and Steel Institute.

Nickel Steel.—The use of nickel steel is increasing in structural work and railroad curves, while also many castings have been made during the year containing the usual 3.5 per cent. of nickel. The large heats of the open hearth process are an obstacle to the manufacture of alloy steel castings, because either the whole heat, or none, must be of alloy steel. Therefore, the use of nickel steel castings, in spite of this hindrance, is a sign of the times, and is one of the reasons of the extension of small converters for casting work.

Nickel steel wire has also been tried a good deal during the year for cables and other purposes, but it is too early yet to make an intelligent comparison between it and wire of carbon steel. The International Nickel Company has vigorously advocated the making of boiler tubes containing 25 to 30 per cent. of nickel, which are almost non-scaling, and these have been adopted somewhat, though not as much in the United States as abroad.

Automobile Parts.—Important parts of automobiles are now frequently made of alloy steels, especially nickel steels, chrome steels, and nickel-chrome steels. Some of these are used after tempering, or after a double heat treatment. This subject is well exploited by one highly qualified to speak with authority, and the reader is referred to an article by Dr. Léon Guillet in the *Journal* of the Iron and Steel Institute, No. 2, 1905, pp. 166 to 189, on "Steel Used for Motor-Car Construction in France."

Research.—This year has been productive of much research in alloy steels, and reference on this subject should be made to the issues of Le Chatelier's periodical, *Revue de Metallurgie.* The present tendency is to search in the field of quaternary alloys; i. e., steels composed of iron, carbon and two other constituents, such as nickel and manganese.

The scientific research, although vigorously prosecuted, has not kept pace with the advance in the manufacture of alloy steels, and we are still making alloys having valuable peculiarities the reasons for which are not yet understood. Important advances have been made in the scientific side, and the reader is referred to the papers in the *Journal* of the Iron and Steel Institute for the year by R. A. Hadfield, F. Rogers, J. O. Arnold and A. McWilliam, L. Guillet, L. Dumas, C. Benedicts, G. B. Waterhouse, G. Dillner and A. F. Enström.

Iron-Carbon Alloys.—In connection with this subject it may be well to note here that the nature of those constituents of the iron-carbon alloys which are abnormal in the cold is now known with a certain degree of positiveness. The theory originally advanced by Osmond, and tentatively adopted by Howe, is strengthened; namely, that austenite is the undecomposed solid solution of iron and carbon, and that martensite, troostite and sorbite are steps in the transition between austenite and pearlite. There still exists, however, a disagreement as to the difference between austenite and martensite.

Miscellaneous.

The increased use of electric power in steel works is worthy of note[1], and especially the use of magnets for handling such unwieldy materials as pig iron, tin scrap[2], etc. In the steel works there seems to be an awakening of interest also in the fluid compression of ingots, and, in general, the avoidance of piping and segregation, while the attention paid to the finishing temperatures in rolling and to the crystalline structure of rolled and forged material is unabated[3].

The number of microscopic laboratories has been increased, and this method of testing has now come to be almost universally employed.

The year has witnessed an important extension in the employment of concrete reenforcement, and of the styles and shapes of steel material used for the same. Manganese steel is steadily advancing in the amount in which it is used for railroad rails on curves, especially elevated and street railroads, and for burglar proof safes.

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[1] In *Electrician* (London), March 2, 1906, descriptions of some of the installations will be found.

[2] See *The Foundry*, Vol. XXVI, April, 1905, p. 83; *The Electrochemical and Metallurgical Industry*, Vol. III, July, 1905, p. 278, "Lifting Electromagnets in the Iron and Steel and in the Tin Industry." See also *Machinery*, March, 1905.

[3] "Piping of Steel Ingots," by N. Lilienberg, *Trans.* of the American Institute of Mining Engineers, Feb., 1906. See also *Stahl und Eisen*, Vol. XXV, pp. 832, 865 and 923; "Segregation in Steel Ingots," by Benjamin Talbot; *Journal* of the Iron and Steel Institute, No. II, 1905, pp. 204 to 223

LEAD.

By W. R. Ingalls.

The total production of lead in the United States in 1905 showed a small increase; the production of lead of domestic origin showed also an increase, which was due chiefly to the smelters of the Mississippi Valley, their ore supply being derived principally from southeastern Missouri. The prospect is for a further important increase in the production of lead from this source during 1906. The lead manufacturing industries participated in the general prosperity of 1905, and at the end of the year stocks of metal in the hands of refiners were quite insignificant. The statistics of production and consumption are given in the following tables.

STATISTICS OF LEAD IN THE UNITED STATES
(Short tons.)

Year.	Produced from Domestic Ores.				Imported in Ores and Bullion.	Total Production and Imports.	Exported in all Forms.
	Desilverized.	Soft.	Antimonial. (a)	Totals.			
1896	135,332	33,428	5,932	174,692	80,159	273,826	50,983
1897	144,649	45,710	7,359	197,718	92,117	302,859	60,353
1898	169,364	50,468	8,643	228,475	89,209	348,845	78,168
1899	171,495	40,508	7,377	217,085	76,423	317,196	74,944
1900	221,278	47,923	9,906	279,107	114,397	425,824	100,288
1901	211,368	57,898	10,656	279,922	112,471	458,033	100,026
1902	199,615	70,424	10,485	280,524	107,715	458,456	82,228
1903	188,943	78,298	9,453	276,694	106,407	418,601	81,971
1904	200,858	90,470	10,876	302,204	112,852	415,056	84,142
1905	205,665	105,623	11,186	322,474	98,378	420,852	59,741

(a) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin. In 1905, the first year in which the separation has been possible, the antimonial lead from foreign sources amounted to 2730 tons.

METALLURGICAL PRODUCTION OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Domestic Origin.						Foreign Origin.		Grand Total.
	Desilverized	Antimonial	S. E. Mo.	S. W. Mo.	Miscel.	Total.	Desilverized	Antimonial	
1905	205,665	8,456	81,299	21,324	3,000	319,744	83,504	2,730	405,978

The production of refined lead in the United States in 1906 was 405,978 short tons, of which 319,744 tons was of domestic origin, and the remainder

IMPORTS OF LEAD IN ORE, BASE BULLION, PIGS, BARS AND OLD. (a)

Source.	1901.	1902.	1903.	1904.	1905.
	Short tons.	*Short tons.*	*Short tons.*	*Short tons.*	*Short tons.*
United Kingdom	201.3	396.3	776.4	247.3	795.0
Germany	335.6	476.4	704.9	365.6	125.1
Other Europe	1.2	671.1	225.7	82.8	58.8
Canada	26,065.0	9,732.5	9,600.4	8,951.9	8,181.5
Mexico	81,726.8	93,742.3	93,068.3	102,903.0	87,583.8
South America	4,108.7	2,690.1	1,947.8	290.0	1,577.2
Other Countries	32.5	6.1	83.2	11.0	56.3
Total	112,471.1	107,714.8	106,406.7	112,851.7	98,377.7

(a) Refined lead, i. e., in pigs, bars and old, is a small part of the total. It was, in 1901, 604 tons; 1902, 2,529 tons, 1903, 3,023 tons; 1904, 8,724 tons; 1905, 5,720 tons.

of foreign origin. Of the domestic production, 205,665 tons was desilverized lead; 8456 tons was antimonial lead; and 105,623 tons was lead from Missouri, Wisconsin and other Eastern States, commonly referred to as "soft lead." These statistics are based upon reports received from all the refiners, with the exception of one small concern whose output we have estimated at 2000 tons.

The statistics of lead production which are published in The Mineral Industry are metallurgical, i. e., they represent the output of lead in final marketable form. The refiners in making their reports separate their output according to origin, domestic or foreign, so far as possible, but absolute precision in this respect is seldom possible, wherefore the figure of most accuracy is the grand total. The statistics of crude lead production collected by States, in a different manner, show a total for 1905 of 320,256 tons. The details of the latter estimate are given on a subsequent page.

In the table of refined lead production, we adhere to the old classification of "desilverized" and "soft" lead, although this is inaccurate. Desilverized lead is soft lead, in fact, is much softer than a good deal of the Missouri lead, produced and sold as "chemical hard," but entered statistically as "soft" lead. Moreover, a considerable proportion of the Missouri lead is desilverized, this being the only way whereby some of it can be converted into a high grade corroding metal; incidentally, the small silver content of this lead is recovered. A small portion of the non-argentiferous lead ore produced in the Mississippi Valley is sold to silver-lead smelters, who turn out the product in the form of desilverized.

It is impossible to compute accurately the consumption of lead, because reliable statistics as to stocks on hand cannot be obtained. The stocks of lead in the hands of refiners are now much smaller than formerly was the case, it being the policy of the largest producing interest to compel the consumers to carry the stocks, which result is accomplished by a special method of sale. It was well known that at the end of 1905 the stocks of lead in the hands of refiners were very small, immediate deliveries of metal then com-

CONSUMPTION OF LEAD IN THE UNITED STATES.

	1904.	1905.		1904.	1905.
Supply:			Deduction:		
Total production desilverized	299,788	289,169	Re-exports of foreign	83,887	59,456
Domestic production soft lead	90,470	105,623	Exports of domestic lead	35	63
Production antimonial lead	10,876	11,186	Stock, domestic lead, Dec. 31	10,000	4,,000
Imports foreign refined lead	8,724	5,720	Stock, foreign in bond, Dec. 31	11,481	8,204
Stock, domestic lead, Jan. 1	21,000	110,000			
Stock, foreign in bond, Jan. 1	10,694	11,481	Total deductions	105,403	71,723
Total supply	441,552	433,179	Apparent home consumption	336,149	361,456

manding a premium. We have estimated the stocks on hand, Dec. 31, 1905, at 4000 tons, which figure, about 1 per cent. of the production during

the year, is to be regarded as merely nominal. The consumption of lead in 1905 showed a large increase, there being not only an increase in the domestic production, but also a decrease in stocks, and a large falling off in the exports of lead of foreign origin; i. e., foreign lead brought into this country for refining in bond was withdrawn from bond for domestic consumption, this being equivalent to a direct importation of the metal.

The subject of antimonial lead is discussed under the caption "Antimony," to which article reference should be made. This grade of lead was in great demand in 1905 and commanded a premium of 1 @ 1.5c. per lb. over the price of desilverized. Formerly it used to sell at a discount.

PRODUCTION OF LEAD BY STATES.
(In tons of 2000 lb.)

State	*a* 1901	*a* 1902	*b* 1903	*b* 1904	*d* 1905
Colorado..........	73,265	51,833	43,276	49,290	*f*57,856
Idaho.............	79,654	84,742	94,611	103,411	*h*107,000
Missouri-Kansas....	*c*67,172	*c*79,445	*c*86,439	*c*92,119	*g*102,500
Montana..........	5,791	4,438	3,138	3,454	*e*3,500
Utah.............	49,870	53,914	48,573	53,647	44,500
Others............	8,452	6,425	6,365	5,283	*e*4,900
Totals..........	284,204	280,797	282,204	307,204	320,256

(*a*) Statistics of the U. S. Geological Survey, representing "lead content of ore smelted." (*b*) Statistics of the U. S Geological Survey, representing production of merchant lead. (*c*) Includes also the production of Wisconsin, Illinois, Iowa, Virginia and Kentucky. (*d*) Production of crude lead, distributed according to State of origin. (e) Estimated. (*f*) Report of State Commissioner of Mines. (*g*) Includes 1500 tons from Iowa, Illinois and Wisconsin; this may not represent the total production of these States, anything in excess appearing under the classification of "Others." (*h*) Partly estimated.

General Review of the Industry in 1905.

The year was characterized by further consolidations of producing interests, all tending to increase the hold which the American Smelting and Refining Company has upon the industry. That company, through the American Smelters' Securities Company, absorbed the Guggenheim Exploration Company and its various interests, together with certain other interests that were outside of it. The American Smelting and Refining Company, through the American Smelters' Securities Company, now controls all of the lead smelting and refining plants on the Pacific coast, including the famous old Selby works, near San Francisco. It also controls the Federal Mining and Smelting Company, which is one of the largest producers of silver-lead ore in the Cœur d'Alene. That company controls the Sullivan Group Mining Company, which has a mine and smelter at Marysville, B. C., the bullion from which is shipped to Everett, Wash., for refining. Through the Federal Lead Company it controls a large smelting works at Alton, Ill. The Federal Lead Company further increased its interest in the Flat River district, Missouri, by the purchase of the Central mine, and by making contracts with other mining companies for the smelting of their ore.

At the present time there are only four large producers of refined lead in the United States. These are the American Smelting and Refining Com-

pany through itself and its allied interests, the Balbach Smelting and Refining Company, the St. Joseph Lead Company and the St. Louis Smelting and Refining Company, the last being one of the subsidiary companies of the National Lead Company. It is even probable that there will be a further consolidation of interests, since during the latter portion of the year there was a good deal of talk as to the eventual consolidation of the United Lead Company (American Smelting and Refining Company) and the National Lead Company, a combination which has been under consideration for several years. This has since been consummated, the National Lead Company absorbing the United, and itself entering into certain trade agreements with the American Smelting and Refining Company.

Of the outside smelting interests, the United States Mining Company, which has a plant near Salt Lake City, is now the most important. This company derives its ore partly from its own mines in the Bingham district, and partly from small mines in other districts. During 1905 this company secured a controlling interest in the old Richmond and Eureka mines at Eureka, Nev., which it is now reopening.

Early in 1906 the United States Mining Company was reorganized as the United States Mining, Smelting and Refining Company, with a large increase of capital and the declared intention of entering into competition with the American Smelting and Refining Company. Among its assets are the controlling interest in the De Lamar's Copper Refining Company.

The De Lamar's Copper Refining Company has bought property in East Chicago for the purpose of erecting a lead refinery of about 2500 tons per month capacity. The refinery will be in operation during the second half of this year and will treat the product of the United States Mining, Smelting and Refining Company, and also custom bullion.

Experience has shown that Chicago is a very suitable location for works of this character, it having been an important lead refining center ever since the sixties, and being now the location of one of the important refineries of the American Smelting and Refining Company. The establishment of the new plant will afford healthful competition which will tend to stimulate the erection of small lead-smelting plants in the West, like several which were undertaken last year.

The De Lamar refinery will be of especial metallurgical interest, inasmuch as the Betts electrolytic process is to be employed, and it will in fact be the first electrolytic lead refinery in the United States, although not the first in North America, the process having been in use at Trail, B. C., for several years.

Other producers of base bullion are the Ohio & Colorado Smelting Company, which has a plant at Salida, Colo., and various small works in different parts of the country, which treat chiefly the ores from a single group of mines. Among such plants are those of the Luna Lead Company, at Dem-

ing, N. M., and the Mowry Mines Company, at Patagonia, Ariz., which were constructed during 1905. Other small plants, under construction, will be in operation in 1906, including one at Sandpoint, Idaho, one at Keeler, California, and one at Needles, California. Lead furnaces are also contemplated by the Arizona Smelting Company, at Humboldt, Arizona.

In the metallurgy of lead the great development of 1905 was the introduction of the Huntington-Heberlein process in the plants of the American Smelting and Refining Company. This process has already been introduced at Pueblo, Colo., Salt Lake City, Utah, and East Helena, Mont., and it is understood to be the intention of the operating committee of the company to install it in all the plants of the company. The process was previously introduced at Mapimi, Mex., and at Marysville, B. C. Installations of the process are also being made at Nelson, B. C., by the Hall Mining and Smelting Company, and at Trail, B. C., by the Canadian Smelting Works.

The Huntington-Heberlein process is analogous to the Carmichael-Bradford and Savelsberg processes, which have been designated by the generic name of "lime-roasting of galena." In these processes the sulphide ore, mixed with a certain proportion of limestone and silicious flux, is charged into a large, hemispherical cast-iron pot, usually of capacity for eight to ten tons of charge. Air is introduced at the bottom of the pot and is blown through the charge at low pressure.

An exothermic reaction takes place, the charge becoming red hot, and sulphur dioxide coming off. At the end of the reaction, this extending up to sixteen hours, desulphurization is almost completely effected. The charge is then dumped out of the pot, by inverting the latter, dropping out as a solid, red hot cake. This is then broken up by wedging and sledging to size suitable for the blast furnace, to which it is directly delivered. Owing to its excellent physical characteristics the speed of the blast furnace is greatly increased, the blast furnace can be operated at lower wind pressure and the production of matte is greatly reduced. These advantages combine to effect a very important saving in the treatment of such ore, besides which the loss of silver and lead is greatly reduced, the desulphurization in the converters being effected at a temperature so comparatively low that volatilization of lead and silver is insignificant.

The essential difference among the three processes mentioned above is as follows: In the Huntington-Heberlein the ore is first partially roasted; that is to say, it is burned down to a content of 10 per cent. to 11 per cent. sulphur, and is then charged into the converter; in the Savelsberg process the ore mixed with the proper proportion of limestone and quartzose material is charged directly into the converter, there being no preliminary roasting; in the Carmichael-Bradford process the ore is mixed with a proportion of gypsum, and is then charged directly into the converter. These pro-

cesses have been described in a series of articles which appeared in the *Engineering and Mining Journal* during 1905.

Although the operation of these processes and their results are now well known the precise nature of the reactions which operate in them is by no means understood as yet, and both the experience with them and the data concerning them which have been published are still too meager to lead to a thorough discussion of the subject. One thing, however, is clear: These processes constitute one of the most important improvements in the metallurgy of lead during the last 25 years.

The Huntington-Heberlein process was introduced extensively in Europe in Australia and in Tasmania, before it was tried in the United States. In those countries it has already displaced to a large extent the old reverberatory roasting furnace, and also in Silesia it has displaced the old roast-reaction method of lead smelting.

The right to the use of the Huntington-Heberlein process in the United States is controlled exclusively by the American Smelting and Refining Company.

The domestic production of lead ore continues to come from a limited number of districts, that is to say the bulk of it, although a large number of mines produce a small quantity of lead. The relative importance of the big lead districts has been increasing, however, during the last two or three years, and in 1906 about 80 per cent. of the domestic output was derived from five, while upward of 50 per cent. originated in two, viz. the Cœur d'Alene and the disseminated district of southeastern Missouri. The statistics of the large districts are given in the following table:

District.	Production, tons.				Per cent.				Ref.
	1901	1902	1903	1904	1901	1902	1903	1904	
Cœur d'Alene..........	68,953	74,739	89,880	98,240	24.3	26.3	32.5	32.5	*a*
Southeast Missouri.....	46,657	56,550	59,660	59,104	16.4	19.9	21.2	19.6	*b*
Leadville, Colorado....	28,180	19,725	18,177	23,590	10.0	6.9	6.6	7.8	*c*
Park City, Utah.......	28,310	36,300	36,534	30,192	10.0	12.8	13.2	10.0	*d*
Joplin, Mo.-Kansas....	24,500	22,130	20,000	23,600	8.6	7.8	7.2	7.8	*e*
Totals.............	196,600	209,444	224,251	234,726	69.3	73.7	81.0	77.7	

(*a*) The production in 1901 and 1902 is computed from direct returns from the mines, with an allowance of 6 per cent. for loss of lead in smelting. The production in 1903 and 1904 is estimated as 95 per cent. of the total lead product of Idaho. (*b*) This figure includes only the output of the mines at Bonne Terre, Flat River, Doe Run, Mine La Motte and Fredericktown. It is computed from the report of the State Lead and Zinc Mine Inspector as to ore produced, the ore (concentrates) of the mines at Bonne Terre, Flat River and Doe Run being reckoned as yielding 60 per cent. lead. (*c*) Report of State Commissioner of Mines. (*d*) Report of Director of the Mint on "Production of Gold and Silver in the United States," with allowance of 6 per cent. for loss of lead in smelting. (*e*) From statistics reported by THE MINERAL INDUSTRY, reckoning the ore (concentrates) as yielding 70 per cent. lead.

Developments in the Missouri and Idaho districts are described in excellent articles by Messrs. Wheeler, Zook and Easton, which are published further on. Southeastern Missouri and the Cœur d'Alene both showed important increases in output during 1905, and there is no evidence that either of these important districts is yet anywhere near its maximum productive capacity. In Colorado, Utah and other States of the Rocky

Mountains the production of lead is being helped by the production of zinc ore, for which at present there is a great demand. In the mines of those States zinc and lead are commonly associated, wherefore a strong demand for one kind of ore will naturally increase the production of the other.

Lead Mining in the United States in 1905.[1]

Arizona.—The Mowry Mines Company, operating the old mines of that name at Patagonia, south of Tucson, erected an 80-ton smelting furnace. The ore is carbonate. A concentrating mill is also to be built.

California.—This State has not figured as a producer of lead of any consequence for many years, but between 1869 and 1879, when the mines at Cerro Gordo, Inyo county, were in operation, it used to make a relatively large production. Cerro Gordo was finally abandoned because the mines had become unprofitable under the conditions then existing and since then have been forgotten until recently. They are now being reopened by the Great Western Ore Purchasing and Reduction Company, which is erecting a 150-ton furnace at Keeler. Besides the low-grade ore left from former operations, which it is believed can now be extracted profitably, there are large accumulations of slag produced by the early smelters, which ought to be rich, judging from the contemporaneous accounts of the smelting practice.

Colorado.—The production of lead in this State in 1905 was 57,856 tons, according to the statistics of the State commissioner of mines. Of this, the American Smelting and Refining Company produced 50,000 tons, and the Ohio & Colorado Smelting Company produced 6400 tons. The details of the production by counties are given in the following table in which Lake county corresponds with the Leadville district, Pitkin county with Aspen, Mineral county with Creede, and Eagle county with Red Cliff:

LEAD PRODUCTION OF COLORADO.
(In short tons.)

County.	1903.		1904.		1905.	
	Short tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
Clear Creek	1,726	$146,255	1,981	$170,365	1,631	153,339
Hinsdale	230	19,467	521	44,773	446	41,919
Lake	18,177	1,540,287	23,590	2,028,777	26,424	2,483,875
Mineral	4,300	364,409	6,673	573,897	5,940	558,397
Ouray	1,675	141,964	1,022	87,915	2,674	251,368
Pitkin	1,663	1,409,644	9,441	811,965	10,987	1,032,870
San Juan	3,485	295,280	4,644	399,412	3,223	302,942
Others	4,529	383,817	5,901	507,411	6,531	613,797
Total	50,757	$4,301,123	53,773	$4,824,515	57,856	5,438,507

Leadville, which is the largest producer, enjoyed a prosperous year in 1905, the total tonnage of all kinds of ore being the largest in the history of the district. The ore averages low in grade, however, and the pro-

[1]Reference should also be made to the article on "Zinc," the districts producing both lead and zinc, in which zinc is the more important, being described under that caption.

portion of good lead ore is comparatively small. The rich carbonates, which were produced so largely in the '80's, have long since been practically worked out. The diversity of ores produced at Leadville, and the large tonnages, continue to make the camp the great resource of the Colorado smelters, and one of the largest plants of the American Smelting and Refining Company—the Arkansas Valley works—is situated at Leadville. This plant is constantly being enlarged and improved; of all the many smelting works that have been run at Leadville, it is now the only one in operation.

According to the Leadville *Herald-Democrat*, the production of the various classes of ore in 1905 was as follows: Carbonate, 86,174 tons; iron ore, 127,170; sulphide ore, 297,909; zinc ore, 159,747[1]; silicious ore, 154,370; manganese ore, 6000; total, 831,370.

The production of base bullion derived from these ores is estimated at 27,000 tons, which would correspond to 26,424 tons of refined lead. Referring this figure to the tonnage of carbonate and sulphide ore, the average appears to be about 7 per cent. lead.

The largest producers of ore are the Western Mining Company, the Iron Silver Mining Company and the Yak Mining and Milling Company.

Aspen continues to produce considerable lead by virtue of the large tonnage of silver ore, containing a little lead, which is obtained from its mines. The ore shipments in 1905 were larger than for several years previous. The Smuggler is the principal producer.

Outside of Leadville and Aspen, the largest lead producing district of Colorado is Creede, the lead ore of which is produced largely in the form of concentrate, assaying 65 per cent. and upward in lead. The year was fairly prosperous. In this camp, as in others, consolidation of interests has been the order of the day. The most important properties are now operated by the United Creede Mines.

Idaho. (By Robert N. Bell.)—This State continues to lead all the others in lead production and takes high rank also in silver production. The remarkable increase in quantity and grade of the ore recently shown in some of the deepest development of the leading Cœur d'Alene producers, and other Idaho resources, seems likely to insure the present position of the State for some time to come.

While the Cœur d'Alene district, in Shoshone county, was the origin of over 95 per cent. of the total lead output of the State in 1905, estimated at 130,000 tons,[2] three other counties contributed the remainder, viz., Blaine, Lemhi and Custer, lying in the east-central part of the State, and their different districts contain many interesting deposits, geologically, that are

[1]With respect to this figure reference should be made to the article on "Zinc" further on in this book.

[2]This figure is for the gross lead content of the ore, without allowance for loss in smelting. Even after making allowance for loss in smelting, it is impossible to reconcile this figure with the reports received directly from the smelters. Our estimate of the lead production of Idaho in 1905 is 107,000 tons.

likely to afford an important and increasing output. Their present development is retarded by lack of railway transportation facilities. In addition to these counties, there are three more in the central and southwestern parts of the State, viz., Boise, Washington and Owyhee counties, that contain deposits of low grade lead ore, which may, in time, develop extensive resources.

Almost all the Idaho silver-lead ore deposits are found associated with metamorphosed sedimentary formations and deep fissuring, and in all the districts south of Shoshone county limestones and igneous intrusive rocks are conspicuously developed, and copper is generally associated with the ores.

At the Gilmore, in Lemhi county, near the famous old Viola mine, a promising deposit of high-grade lead carbonate ore, contained in a nearly vertical fissure in blue limestone, was operated successfully during the summer; 2500 tons of ore were shipped, mostly crude mineral, just as it came from the stopes, that averaged about 50 per cent. lead and 20 oz. of silver. This property is situated in an extensive lead-bearing district that compares closely in ores and formations with the Tintic district in Utah. It is attracting considerable attention, and several new development enterprises have been started there. It is, unfortunately, 75 miles from a railroad, the nearest point being Dubois on the Oregon Short Line railway, but the distance is covered over a smooth, hard, desert road, at a wagon-haul charge of $10 per ton for the ore.

Cœur d'Alene. (By Stanley A. Easton.)—The yield of the mines of this district during 1905 was greater than that of any previous year. The work of most importance was the exploration and exploitation of the deepest levels from which all of the older mines are now producing wholly. These levels are scarcely to be considered deep, as modern "deep mining" is now accounted; but they are deep in comparison with the work done heretofore in this district. Particular reference is had to the orebodies disclosed by the deepest workings of the Federal Mining and Smelting Company's Mace mine (the consolidation of the old Standard and Mammoth mines); to the deep development on the Morning at Mullan; to that in the Bunker Hill & Sullivan at Wardner; and to that in the Hecla mine at Burke.

Not only are these finds of ore in the deepest known portions of the mines important to the properties themselves, but also they are, in a broad sense, important to the entire district. They show that the ore-bearing depths of these wonderfully productive Cœur d'Alene quartzites are not yet sounded, and that the work is now entering a horizon really yielding an ore of higher grade than that from the upper levels. This excludes, of course, those extreme upper portions, enriched by secondary agencies, where were found the very high-grade orebodies worked in the early days of Wardner; of such enrichment, the upper portions of the Hercules mine near Burke

form a notable example. There were great masses of "secondary" carbonate and sulphate of lead, with a little lead phosphate, yielding, in large shipments, at the rate of 60 per cent. lead and 100 oz. of silver per ton.

These surface orebodies have disappeared completely from all the older mines. The present and the future depend upon the "primary" galena; this is being mined in the bottom levels, as clean from foreign sulphides, as fruitful in silver, and as high in lead, as the best has ever yielded.

In the Bunker Hill & Sullivan, at Wardner, what is possibly the finest body of galena ever found in this district is being worked, together with other valuable orebodies, 2100 ft. below the surface and 3000 ft. from the apex of the vein along its dip; while equally great depths are reached by the workings of the orebodies at Mace, Gem and Mullan.

Development work upon prospects and unproductive properties was conducted with great activity, but no new producer of considerable tonnage has been found. The increase in yield over former years comes solely from the old mines. The number of small producers which have shipped during the year gives hope that several will, in time, reach a productive stage that will rank them with the old and well-known mines. Among properties in the development and shipping stage are the Chesapeake & Tamarack; 16 to 1 mine; and California, on Nine Mile creek; the Senator Stewart, at Wardner, which seems to be the link on the Wardner vein between the mines at Wardner and those on Government Gulch; two or more properties tributary to Murray and now without the transportation facilities to ship either regularly or economically; also on Pine creek, below Wardner, much prospecting and investigating has been done.

The sale of the Morning mine, by Larson & Greenough, to the Federal Mining and Smelting Company, for $3,000,000, was the biggest transaction of the year. A bond was given by its owners for a $\frac{15}{32}$ interest on their holdings in the Hercules to Duluth people.

Water for power was scarce throughout the year; and for this reason there was a regularly increased use of the electric power transmitted from Spokane Falls, for general mine- and mill-work. All the mines are now drawing on the system, consuming about 28,000 horse-power. A high class of efficient labor is employed. There are 2500 engaged directly in the mine- and mill-work alone. Industry and good order prevailed.

The production of the Cœur d'Alene for 1905 was approximately 124,000 tons of lead and 6,690,000 oz. of silver.[1]

Iowa.—See WISCONSIN.

Kansas.—See MISSOURI.

Missouri and Kansas.—The most important lead producing portion of these States is the disseminated district of St. François and Madison counties, Mo. Next in importance is the Joplin district, chiefly in

[1]These figures are for the gross contents of the ore, without allowance for loss in smelting See foot-note on p. 365

southwestern Missouri, which extends over into Kansas. A considerable output of lead is made elsewhere in Missouri, chiefly from the reticulated deposits and fissure veins of the southeastern part of the State. According to the reports of the State mine inspector, the eastern part of the State produced 103,900 tons of ore in 1903, of which 99,434 tons came from St. François and Madison counties; in 1904, those counties produced 98,507 tons out of a total of 103,682 tons. The outside mines afforded, therefore, 4466 and 5175 tons respectively. Their ore is of exceptionally high grade, and being now shipped largely to modern smelteries, may be estimated as yielding 75 per cent. of lead, corresponding to about 3350 and 3750 tons respectively.

The production of the Joplin district in 1905 was 31,679 tons of ore, which may be estimated as yielding 70 per cent., or 22,175 tons of lead. This ore is chiefly smelted locally. There are three smelters in the district, all employing Scotch hearths. Their output in 1906 was 21,324 tons. It is merely a coincidence that this figure agrees so closely with the computed lead content of the ore produced, a portion of the latter being used for the manufacture of sublimed white lead, while moreover a small quantity of ore is obtained from outside sources.[1]

The lead production of St. François and Madison counties in 1905 was about 80,000 tons. The ore is smelted chiefly by the Federal Lead Company, at Alton, Ill., the St. Louis Smelting and Refining Company at Collinsville, Ill., and the St. Joseph Lead Company, at Herculaneum, Mo. The first two use the Scotch hearth-bag house process, and besides smelting the ore of their own mines, purchase considerable supplies from others. The St. Joseph Lead Company, which roasts in reverberatory furnaces and smelts in blast furnaces, treats also the ore of the Doe Run Lead Company. Further progress was made during the year in replacing the old circular furnaces by larger, rectangular ones, of the Colorado type. The Central Lead Company smelts in blast furnaces, but the plant was closed in the early part of 1906, the ore being sent subsequently to the Federal smelter. Con-

St. Francois and Madison counties	80,000	tons
Other southeastern counties	3,500	"
Missouri—Kansas (Joplin district)	22,175[1]	"
Grand total	105,675	"
Deduct for lead in pigment	4,650	"
Pig lead	101,025	"

[1]Including lead in sublimed white lead, but not including lead in zinc smelters' residues.

siderable of the Desloge ore was also shipped to other smelters, but some of its reverberatory furnaces were operated. The Mine La Motte Company has a blast furnace smelting plant, which was operated. The production of all the above mentioned smelters in 1905 was 81,299 tons. The Penn-

[1]The estimated yield of 70 per cent. is doubtless somewhat low. The ore of the district averages about 78 per cent lead. An extraction of 90 per cent. would correspond with a yield of 1404 lb. per ton, i. e. 70 per cent. The advantage gained over that figure depends upon the efficiency of the fume collecting system of the smelters. One which has a bag-house effects of course a much higher percentage of saving.

sylvania Smelting Company of Carnegie (near Pittsburgh, Penn.) obtains occasional lots of ore from Missouri.

The total lead production of Missouri and Kansas in 1905 was approximately as given on page 368.

Southeastern District. (By H. A. Wheeler.)—The output of the southeastern Missouri lead district in 1905 approximated 82,000 short tons; of this, St. François county (or the Bonne Terre and the Flat River districts) produced about 90 per cent.; Madison county (or the Fredericktown district), including Mine La Motte, produced about 7 per cent.; and the numerous small mines scattered throughout the adjoining Washington, Jefferson and Franklin counties produced about 3 per cent. This is by far the largest output in the history of this district, as it exceeds the production of 1904, the previous high-water mark, by nearly 12 per cent.

The year was free from labor troubles, although the Miners' Union threatened to start another strike in the Flat River district in the summer, which was fortunately averted by concessions on both sides.

Prospecting was inaugurated on a scale never before attempted. Hitherto, from five to ten diamond drills were to be found scattered throughout the district or on the frontier, searching for new orebodies. These drills were usually operated by prospecting companies, who rarely gave the land a fair trial, as usually only a few random holes were put down, and much ground was hastily condemned on meager and insufficient data. In 1905 it was the old companies that were vigorously carrying on a search for new orebodies and attempting to extend the area of the lead belt. Over 30 diamond drills were prospecting, and on lands that are beyond the known limits of the recognized lead belt.

An innovation in prospecting was the payment of $3 per acre for the privilege of "optioning" the land for a year while carrying on the diamond drilling. Hitherto a large advance payment on the price for the land was exacted on a spot-cash basis for giving a drilling option for 6 to 12 months; but now the land-owners are practically renting their land for double their former rentable value as farms, while the option is held on the land, besides getting a large advance over a quick-sale valuation.

The minimum price that is asked for lands that are miles beyond the known lead belt is $100 per acre, while more favorably situated lands that are nearer known developments are being held at $200 to $400 per acre. These prices are not regarded as high; for if a good orebody is found, the lead-bearing ground is likely to produce from $50,000 to $500,000 worth of lead per acre.

This vigorous prospecting is sure to open up new orebodies, though it will be two or three years before the new discoveries can be developed and equipped into large producers.

An important event in 1905 was the further concentration of New York

interests in this district by the sale of the Central Lead Company's property to the Guggenheim interests. This was one of the few southeastern Missouri lead properties owned by the St. Louis parties. The new management will consolidate it with the adjoining properties—the Derby and Federal—and make it one of the finest properties in the district. A central power and milling plant is to be erected, from which the three mines will be operated; new shafts are to be sunk on the Central; and the old mine, which was flooded during the strike two years ago, is to be pumped out.

The Columbia mine, at Flat River, continued shut down, and through internal dissensions and lawsuits is likely to remain non-productive for some time.

The Mine la Motte property, in Madison county, has been taken out of the hands of the promoters who sold its watered stock at high and unwarranted figures. The property has been bonded for $300,000, and the proceeds from the sale of the bonds are to be devoted to improving its lead-producing capacity.

The St. Joseph Lead Company, of Bonne Terre, materially increased its plant and output during 1905, and it is sinking a new shaft, No. 14, on its Bonne Terre land. This company has a large acreage of unprospected land under option, on which it is operating 20 diamond-drills. In addition to paying its usual 6 per cent. dividend, it declared a 25 per cent. stock dividend last May.

The St. Louis Smelting and Refining Company, at Flat River, installed a couple of electrically operated four-stage centrifugal pumps in one of its mines; this is an innovation in this district, where the water has hitherto been handled by steam pumps.

The Doe Run Lead Company, at Flat River, installed a central electric power plant at its Flat River property last summer, and made considerable improvements in its mill at Doe Run. A large amount of prospecting is being conducted by this company, especially in the outlying districts.

The Desloge Lead Company, at Desloge, had an active, prosperous year; this is the last of the large companies operating in this district that is controlled by St. Louis interests.

The Madison Lead Company, at Fredericktown, made improvements in its mill and mine during 1905, and sunk a new shaft. The North American Lead Company, also at Fredericktown, proposes to increase its output materially in 1906. This and the Madison property are controlled by Ohio interests.

The Union, Manhattan, Penicaut and Elizabeth properties in St. François county have not yet been put on a productive basis; their absorption by a new company is said to be contemplated, as they have considerable merit, and could be made important producers if equipped with modern machinery. The Irondale lead mine, at Irondale, is still full of water; the

Renault property, in Washington county, since the death of its manager, has been operated on only a limited scale.

Southwestern Missouri. (By Jesse A. Zook.)—The lead ore of the Joplin district being mined in connection with zinc ore, which is by far the more important of the two, the industrial record of the district in 1905 is given under the caption "Zinc" further on in this volume. The production of lead ore was 31,679 tons. The statistics of production since 1893 are as follows:

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1894....	32,190	1897....	30,105	1900....	29,132	1903....	28,656
1895....	31,294	1898....	26,687	1901....	35,177	1904....	34,362
1896....	27,721	1899....	23,888	1902....	31,625	1905....	31,679

Lead concentrate began in 1905 at $60 per ton; advanced the following week to $63, from which it declined to $57 on the first of February; then up to $59 at the end of the month; $58 the first half of March, continuing upward to $61.50 in the first half of May; down to $57 on the first of June; up to $60 on the first of July; to $61.50 on the first of August, and to $64 at the end of that month. Throughout September the price was uniform at $63.50, but increased through October until on the first of November $66 was paid. The next week it was $70; the third week $71.50; and the last week of November $74 per ton was paid for choicest grades. During December prices advanced $1 to $2 per ton each week, closing the year at $80.

The highest price paid during 1905 for lead concentrate was $80 per ton in December; the lowest price was $57 per ton at three periods of the year, occurring one week each in February, May and June. The average of the year for all grades of ore was $62.12 per ton. The statistics of price since 1898 are as follows:

Year.	1899	1900	1901	1902	1903	1904	1905
Highest.	$55.00	$56.50	$47.50	$50.00	$60.50	$62.00	$80.00
Average.	51.34	48.32	45.99	46.10	54.12	54.80	62.12

Nevada.—The lead production of this State in 1905 was insignificant. A little ore produced by the Rocco-Homestake mines at White Pine and other small producers was shipped to Salt Lake City for smelting. It is probable that the lead production of this State will soon show an increase, inasmuch as the famous old mines at Eureka are now being reopened. The Eureka and Richmond companies were consolidated in a new company, known as the Richmond-Eureka Mining Company, which is controlled by the United States Mining, Smelting and Refining Company. It was the common report, when the mines were closed down

about 1893, partly because of the fall in price of silver and partly because of existence of the old hostility between the two companies, which precluded the possibility of joint action as to pumping, etc., that there were good prospects for ore in the lowest levels. This is now to be investigated. It is contemplated also to ship the old slag dumps to Salt Lake for smelting.

New Hampshire.—A mine has been opened at Woodstock. It has been equipped with a concentrating mill, which will be operated in 1906.

New Mexico.—The Luna Lead Company has erected a smeltery at Deming, Luna county, and is purchasing lead-silver ores in the open market(1906). The bullion produced at the smelter is consigned to the Colwell Lead Company, of New York. The Deming smelter is an independent enterprise, and will be of great assistance to the mines of the region.

(By Charles R. Keyes.)—The mining of lead ore in New Mexico increased in 1905. The smelter at Deming stimulated mining in its vicinity. In the Granite Gap district, in Grant county, new developments were made; the camp is now becoming an important producer. Another locality coming into prominence as a lead sulphide camp is in the Caballos district, in Sierra county. The ore occurs in well-defined veins. At the present, a large concentrating plant is being installed and a smelter is being provided. The camps in the Organ mountains and in the Black range showed increased activity. In the Magdalena camp large sulphide bodies have been developed. The sulphide ores are a mixture of lead and zinc. A concentrator will soon make a rough separation of the former, which will be smelted on the spot. New lead stopes have also been found at the top of the range; they promise important developments.

Utah.—The lead production of Utah in 1905 is estimated at 44,500 tons, against 53,647 tons in 1904. The set-back appears to have been due, however, only to temporary causes, chiefly the accident to the Ontario tunnel at Park City. The principal producing district was Park City, as heretofore. The other important lead districts of Utah are Bingham, Little Cottonwood, Stockton and Frisco.

At Park City, the cave-in of the Ontario drain-tunnel, in March, restricted production, the tunnel remaining closed until November. This interfered also with the plans of the Daly-West to develop its orebodies at a depth of 2100 ft. through an extension of the tunnel. The Daly-West and the Silver King were the principal producers of lead ore. The former paid $432,000 in dividends; the latter, $1,200,000.

The silver-lead mines of Park City, and also Frisco have become producers of zinc ore. These developments are referred to under "Zinc."

Important developments were made in 1905 at Alta, at the head of Little Cottonwood Cañon, and it is thought that this famous old camp may regain somewhat of its large production in the early seventies

Large orebodies were developed in the Columbus mine. The Continental Mines and Smelter Corporation (Continental-Alta mine) secured a lease of the right of way owned by the Rio Grande Western and will in 1906 reconstruct the road from Wasatch to Bingham Junction. The Alta Consolidated Mining Company was organized by a Salt Lake and Milwaukee syndicate to take over the old Flagstaff and Emma mines.

The output of Stockton decreased, the Honerine company having cleaned up its old dumps and closed its mill, pending completion of the drainage tunnel. The New Stockton Mining Company, which operates a 60-ton mill, developed some extensive bodies of high-grade ore, and a large tonnage of mill ore.

At Frisco, the Horn Silver is still the most important producer of lead ore. Some interesting bodies of lead ore were developed in the Harrington & Hickory mine of the Majestic Copper Company. The discovery of an extension of the Horn Silver lode by the Frisco Contact Company is regarded as an important event. A complicated faulting of the vein had previously deceived everyone as to the location of the extension.

In Bingham Cañon the lead ores of the Old Jordan, Telegraph and other mines have been neglected recently in the development of their copper resources, but attention is now being redirected to them. This ore used to be produced in large quantity in the form of galena concentrate.

The lead production of Tintic showed an increase, but not enough to make up for the shortage in the outputs of Park City and Stockton.

As in Colorado the lead smelting business in Utah has become very much concentrated. All of the old works are dismantled. There are now two smelters in the State, viz. the Murray plant of the American Smelting and Refining Company, eight furnaces, and the plant of the United States Mining, Smelting and Refining Company, three furnaces. Both these plants draw ore supplies from Idaho and Nevada, as well as from Utah.

The Daly-West company, of Park City, produced 129,347 tons of ore in 1905, of which 18,554 tons was shipped as first grade, and 6366 tons was hand-sorted, leaving 104,008 tons to be crushed and concentrated. This yielded 16,198 tons of concentrate. The milling ore averaged 4.5 per cent. lead (wet assay) and 11.29 oz. silver per ton; the concentrate assayed 28.76 per cent. lead, and 52.56 oz. silver per ton; the tailing carried 0.045 per cent. lead, and 3.3 oz. silver per ton, showing a theoretical saving of 99 per cent. of the lead, and 70.77 per cent. of the silver, although, on the basis of sales, the actual saving was 99.5 per cent. of the lead, and 72.5 per cent. of the silver. The total metal content of the ore and concentrate shipped by the Daly-West during 1905 was as follows: Copper, 1,225,731 lb.; lead, 16,772,978 lb.; gold, 1397 oz.; silver, 1,798,628 oz.; zinc,

9,026,517 lb. The total price received for shipments during the year was $1,234,459. On the basis of one ton of ore mined, the operating costs for the year were as follows: Mining, $3.36; ore handling, $0.25; milling, $1.13; general expenses, $0.40; total, including development, $5.24, showing a net profit for the year of $4.32 per ton of ore mined.

The report of the Horn Silver Mining Company for the year ending Dec. 31, 1905, states production of 3,464,007 lb. lead, 6639 lb. copper, 6,035,553 lb. zinc, 104,231 oz. silver and 335 oz. gold. The ore sales aggregated 4733 tons of crude, averaging $12.44 per ton, 8445 tons of first-class zinc ore at $10.87 per ton, 1349 tons lease ore at $13.97 per ton, 5 tons copper ore at $31.12 per ton, and 258 tons concentrate at $13.88 per ton. The total cost of ore extraction was $62,745.31, an average of $3.85 per ton. The concentration costs were $3,111.50, or $2.13 per ton, and the ratio of concentration was 5.2:1.

Virginia.—The mines at Austinville, Wythe county, now owned by the Bertha Mineral Company, were the only producers in 1906. Their output was 89 tons of lead concentrate. The Albemarle Zinc and Lead Company, at Fabers, Nelson county, developed a promising prospect, and erected a mill, but did not ship any concentrate. Some prospecting was also done in the Rye Valley district in Smyth county, but no deposits of importance were discovered.

LEAD PRODUCTION OF THE WORLD. (*a*)
(In metric tons.)

Year.	Australasia	Austria.	Belgium.	Canada.	Chile.	France.	Germany.	Greece.	Hungary.
1896......	30,000	9,769	17,222	10,977	594	8,232	113,792	15,180	1,911
1897......	22,000	9,860	17,023	17,698	370	9,916	118,881	16,468	2,527
1898......	67,000	10,340	19,330	14,477	13	10,920	132,742	19,193	2,305
1899......	87,600	9,736	15,700	9,917	171	15,981	129,225	19,059	2,166
1900......	87,100	10.650	16,865	28,654	14	15,210	121,513	16,396	2,030
1901......	90,000	10,161	18,760	23,452	455	21,000	123,098	17,644	2,029
1902......	90,000	11,300	19,504	11,478	(*e*) 500	18,817	140,331	15,668	2,243
1903......	141,446	12,162	22,263	9,070	71	(*c*)23,332	145,319	16,093	2,057
1904......	118,979	12,645	23,470	19,000	(*e*) 50	(*c*)18,288	137,580	14,320	2,103
1905......	(*e*)120,000	(*e*)12,500	(*e*)23,000	25,391	(*e*) 50	(*e*)20,000	152,590	10,477	(*e*)2,000

Year.	Italy.	Japan.	Mexico.	Russia.	Spain.	Sweden.	United Kingdom. Foreign Ores.	United Kingdom. Domestic Ores.	United States.	Totals.
1896......	20,786	1,958	63,000	262	167,016	1,530	25,889	31,311	158,479	676,662
1897......	22,407	1,737	71,637	450	189,216	1,480	13,312	26,988	179,369	721,167
1898......	24,543	1,705	71,442	241	198,392	1,559	23,239	25,761	207,271	798,615
1899......	20,543	1,989	84,656	322	184,007	1,606	17,571	23,929	196,938	820,873
1900......	23,763	1,877	63,827	221	176,600	1,424	10,738	24,762	253,204	854,407
1901......	25,796	1,806	94,194	156	169,294	988	19,639	20,361	253,944	892,861
1902......	26,494	1,644	106,805	225	177,560	842	9,113	17,987	254,489	926,870
1903......	22,126	1,728	(*b*)94,181	225	175,109	678	14,900	19,958	276,694	977,412
1904......	23,475	(*e*)1,700	(*e*)103,000	(*e*)225	185,862	589	6,888	19,838	302,204	990,216
1905......	(*e*)25,000	(*e*)1,500	(*e*) 88,000	(*e*)200	(*b*)179,002	(*e*) 500	(*e*) 5,000	(*e*) 20,613	319,744	1,005,567

(*a*) From official reports of the respective countries, except the United States. (*b*) Net exports. (*c*) Commercial statistics of Julius Matton, London. (*e*) Estimated.

Wisconsin—Iowa.—There is a small production of lead in these States, obtained in connection with zinc ore. It is partially smelted locally in Scotch hearth furnaces, one such plant being situated at Dubuque, Iowa, and partially is shipped to the smeltery and refinery of the American Smelt-

ing and Refining Company at Chicago, Ill. The lead production of this region is likely to show an increase, because of the great impetus which zinc mining has received. During the first three months of 1906 the output of galena concentrate in Wisconsin alone was 380 tons.

Lead Mining in Foreign Countries in 1905.

The United States is the largest lead producer of the world. Spain, Germany, Australia and Mexico then rank in the order stated, having in view the production of smelting works. Germany, however, obtains a large part of its ore supply by importation from other countries.

Australia.—By far the major portion of the lead production of Australia is mined at Broken Hill, N. S. W. The ore is chiefly smelted by the Proprietary company, at Port Pirie, and the Sulphide Corporation, at Cockle Creek. Both these companies, as does also the Tasmanian Smelting Company, of Zeehan, Tasmania, employ the Huntington-Heberlein process. A large tonnage of the Broken Hill ore, chiefly in the form of galena concentrate, is exported to Europe. The pig lead production of Australian smelting works in 1904 and 1905 is reported by Julius Matton (in tons of 2240 lb.) as follows:

Works.	1904	1905
Broken Hill Proprietary Co.	68,513	67,062
Sulphide Corporation	23,094	22,246
Smg. and Ref. Co. of Australia	10,599	502
Tasmanian Smelting Co.	*e* 7,800	*e* 9,000
Fremantle Smelter.	5,053	*e* 3,000
Queensland	2,046	*e* 2,000
Totals	117,105	103,810

e Estimated.

The Broken Hill Proprietary produced 5,007,698 oz. silver and 66,462 tons lead in 1905. The total number of men employed at Broken Hill at the close of 1905 was 7717. This is the highest number in the history of the field. In 1900—the boom year—it was 7010. At the end of the year, each of the mines along the line of lode was employing more men than three months previous. The figures for the individual mines were: Proprietary, 3036; Block 10,662; Central, 995; South, 902; South Blocks, 18; Block 14, 170; British, 594; Junction, 157; Junction North, 219; North, 380; others, 385. In addition the Proprietary gives employment to 1361 men at Port coke and 68 at the Iron Knob, 62 at Point Turton, and 75 at the Bellambi Pirie works. At Cockle Creek the Sulphide Corporation finds employment for between 400 and 500 men. This brings the total directly dependent upon Broken Hill mines during 1905 up to over 9500.

In July, 1902, the South Australian Government reduced the freight rate on the Broken Hill ores from Cockburn to the seaboard by 1s. per ton. The Government has now made new rates which took effect Feb. 1, 1906, these being as follows: Crude ores, concentrates, sintered slimes and slime

concentrates—Port Adelaide, 12s. 3d.; Port Pirie, 9s. 9d.; Port Augusta, 10s. 9d.; Wallaroo, 11s. 9d. Zinc tailings, blende concentrates, and slimes, other than slimes concentrates—Port Adelaide, 11s. 6d.; Port Pirie, 9s.; Port Augusta, 10s.; Wallaroo, 11s.

The report of the Proprietary company for the half-year ending Nov. 30 states that the alterations in the smelting furnaces at Port Pirie have been completed, and the recoveries of both lead and silver materially improved, the cost of bullion production being reduced by about 3s. 6d. per ton. Stocks at the close of the year were 4500 tons. Shipments of lead to China, Japan and India amounted to 6250 tons, and the tonnage consumed in the Commonwealth and New Zealand 3050 tons, while the remainder was disposed of in the European markets.

At the mine, the lode on the 1100-ft. level has been proved to be 65 ft. wide, assays giving 17 per cent. zinc, 19 per cent. lead and 13 oz. of silver, which is slightly in advance of the general average of the mine.

The profit per ton of gross ore treated was 12s. 10d.—the highest obtained for eight years. The gross profit for the half-year amounted to £227,299 17s. 8d., which, after deducting £30,340 8s. 5d. for depreciation upon the various plants, left a net profit of £196,959 9s. 3d.

The total output of silver was 2,986,585 oz., as against 2,705,929 oz. for the preceding six months. Soft lead produced was 34,479 tons, against 34,747 tons.

The output of the mine was 309,971 tons of ore, against 296,730 tons during the previous half-year; 94,767 tons mill tailings were treated by the zinc concentration plant as against 41,292 tons. This plant is now capable of handling 6000 tons per week; 738 tons of 100 per cent. sulphuric acid were produced; 44,024 tons of slimes were despatched to the sintering works, and 52,194 tons of sintered material sent down to the smelters. The tonnage of ore treated at the Port Pirie smelters was 15,618 tons less than the previous half-year; the production of refined lead was 309 tons in excess, the grade of ore handled in the two periods being practically similar.

The refinery dealt with 34,692 tons of bullion producing: Silver (fine), 2,481,381 oz.; gold, 1511 oz.; soft lead, 33,504 tons; antimonial lead, 303 tons. The building in connection with the manufacture of spelter is nearly completed, and the first furnace will be built during 1906.

The Broken Hill South Company, during the half-year ended Dec. 31, paid two dividends, amounting to £35,000. A contract has been entered into with the Zinc Corporation for the sale of accumulated tailings, estimated at about 700,000 tons; also for the production for nine years. The Corporation has until March, 1907, wherein to erect plant and start operations. One hundred thousand tons of tailings (the first delivery to be made) has been paid for in cash, and a deposit of £10,000 has also been received, which is to remain with the company during the currency of the contract.

Not less than 200,000 tons have to be taken delivery of or paid for every twelve months.

The mine gives promise of increased productiveness. Operations at the 975-ft. level are being commenced. The new mill, which is being erected near the present plant, is being pushed.

The tonnage and metal contents of the crude ore treated during the last two half-years were:

Half-year ending	Tons.	Assay Value.			Metal Contents.		
		% Pb.	Oz. Ag.	% Zn.	Tons Pb.	Ozs. Ag.	Tons Zn.
Dec. 31	95,269	16.2	6.1	11.9	15,444	585,628	11,344
June 30	96,647	16.9	7.3	12.7	16,365	705,693	12,314

The tonnage, assay value, and recoveries were as follows :

Half-year ending	Tons.	Assay Value.				Metal Contents.			Recoveries.		
		Proportion.	% Pb.	Oz. Ag.	% Zn.	Tons Pb.	Ozs. Ag.	Tons Zn.	% Pb.	% Ag.	% Zn.
Dec. 31	16,420	17.2	68.2	19.1	7.4	11,197	313,848	1,199	72.5	53.6	10.6
June 30	17,201	17.8	69.2	22.2	7.2	11,896	382,421	1,245	72.7	54.2	10.1

The costs per ton of crude ore raised were: Mining, June 30, 1905, 10s. 2.1d.; Dec. 31, 10s. 10.4d.; development, June 30, 8s. 1d.; Dec. 31, 1s. 2.4d. The cost for filling depleted stopes, which is included in the above figures for the last half-year, was 1s. 5.76d. per ton of crude ore raised, against 1s. 5d. for the previous half-year. The concentrating plant treated ore at the following costs: June 30, 3s. 9.4d.; Dec. 31, 4s. 1d. The total mine costs per ton of crude ore were: June 30, 14s. 7.6d. Dec. 31, 16s. 1.8d.; per ton concentrates produced, £4 2s. 2.9d., and £4 13s. 8.6d. The estimated tonnage and average assay value of the accumulated heaps of tailings were: 774,062 tons; assay, 6.6 per cent. lead, 4 oz. silver, 18.2 per cent. zinc; and of slimes, 116,553 tons, assay value, 13.5 per cent. lead, 6.5 oz. silver, 17.8 per cent. zinc. The net profit earned per ton of ore was 14s. 10d., and per ton of concentrates, £4 6s. 6d., against 10s. 10d., and £3 2s. 4d. earned during the preceding six months.

The report of the North Broken Hill Company for the half-year ended Dec. 31, 1905, states that the designs for the new mill to treat 4000 tons per week are now being prepared. The grade of the ore is being well maintained. There is every reason to anticipate good results from the De Bavay process, which is being installed by the De Bavay Treatment Company under a satisfactory contract.

The total quantity of ore raised was 46,301 tons. The mill treated 45,414 tons, assaying 18.65 per cent. lead, 6.9 oz. silver and 14.12 per cent. zinc. The concentrates produced were 8849 tons, averaging 68.98 per cent. lead,

17.35 oz. silver, and 6.83 per cent. zinc, the recovery of the metals being 72.06 per cent. of the lead, 51.79 per cent. of the silver, and 9.42 per cent. of the zinc. The ratio of crude ore treated to concentrates produced was 5.13. Working costs during the term were: Mining, 10s. 8.77d. per ton of crude ore; development, 1s. 10.03d.; milling, 3s. 10.09d.; total, 16s. 4.89d. per ton of crude, or £4 5s. 5.57d. per ton of concentrates. The following table shows the results for the twelve months:

	June 30, 1905. Tons.	Dec. 31, 1905. Tons.
Ore treated	34,689	45,414
Output of concentrates	7,178	8,849
Contents of crude ore:		
Lead	19.31%	18.65%
Recovery	73.33%	71.06%
Silver	7.03%	6.9oz.
Recovery	53.12%	51.79%
Cost per ton of ore	18s. 1d.	16s. 4.89d.
Cost per ton of concentrates	£4 7s. 9d.	£4 5s. 5d.

The Sulphide Corporation, owning the Central mine, has large reserves of ore, about 3,500,000 tons being blocked out above the 800-ft. level. The amount treated annually is about 220,000 tons. It averages 16½ per cent. lead, 18½ per cent. zinc and 11½ oz. of silver. Lead concentrates are produced to the amount of 42,000 tons a year, averaging 60 per cent. lead, 10 per cent. zinc and 29½ oz. of silver per ton. By magnetic concentration about 22,000 tons of zinc concentrates are produced, averaging 40 per cent. zinc, 12 per cent. lead and 13½ oz. of silver per ton.

Bulgaria.—According to a recent report, deposits of zinc and lead ores of considerable extent have been found near Bosilegrad, and their exploitation has already been undertaken by an Italian company. The ores are shipped to Austria and Germany.

Canada.—The production of lead in 1905, according to the statistics of the Dominion Geological Survey, was 27,980 tons[1], against 19,765 tons in 1904. A small part of this is produced in Ontario, but by far the major portion comes from British Columbia, chiefly from the West Kootenay (Slocan and Ainsworth districts), and the East Kootenay (St. Eugene and Sullivan mines). The lead production of British Columbia in 1905, according to the Provincial Mineralogist, was 28,290 tons, against 18,323 tons in 1904. This is produced chiefly in the form of hand-sorted lump ore and mill concentrate, the latter assaying 60 to 65 per cent. lead. The Sullivan Group Mining Company, at Marysville, mines and smelts a heavy sulphide ore, assaying about 27 per cent. lead, 18 per cent. iron, 12 per cent. zinc, and 11 oz. silver. The Huntington-Heberlein process is used. The other smelters of the Province are the Canadian Smelting Works, Trail, and Hall Mining and Smelting Company, Nelson. Both of these are to install

[1]This is a provisional figure; it will be observed that it is a little smaller than the figure reported for British Columbia alone by the Provincial Mineralogist.

the Huntington-Heberlein process in 1906. The Canadian Metal Company, which has erected a zinc smeltery at Frank, Alberta, purchased the Blue Bell mine, and the old Pilot Bay smelting works, on Kootenay lake, and contemplates operating the latter in 1906. The Canadian Smelting Works has an electrolytic lead refinery of 50 tons daily capacity. The Sullivan bullion is exported to Everett, Washington, for refining.

In spite of the high price for lead, which toward the end of the year rose to the figure at which the Government was no longer required to pay bounty on the production, the output of the West Kootenay did not show any gain. As heretofore, the major part of the production was made by the St. Eugene mine. Of the 28,290 tons of metallic lead produced, about 8000 tons were from ore or concentrates exported to Europe, and the remainder was the output of British Columbian lead smelters. A fire at the St. Eugene stopped shipments from that mine for two months, otherwise the Province's largest production, viz., 31,679 tons in 1900, would most likely have been exceeded in 1905. The output was materially augmented by the Sullivan smelter, which went into operation about midsummer.

The St. Eugene's production of silver-lead ore in 1905 was as follows, the corresponding figures for the previous year being given in brackets: Ore milled, 130,000 tons (73,000 tons); concentrates shipped, 30,000 tons (15,000 tons); lead produced, 40,000,000 ℔. (21,000,000 ℔.); silver, 1,000,000 oz. (541,500 oz.) Of the ore and concentrates produced, 11,708 tons were shipped to Europe and the remainder to Nelson and Trail, B. C. Development work consisted of 2029 lin. ft. of sinking and raising and 5744 lin. ft. of cross-cutting and drifting, making a total of 7773 lin. ft. and bringing the aggregate footage of development in the mine up to about 42,000 ft., or 8 miles. The net earnings for the year were about $500,000. Four quarterly dividends each of 2 per cent. were paid, making 16 per cent. in all to date, and representing a total on the issued stock ($3,202,000) of $512,320 of distributed profits.

The amount of the bounty paid by the Dominion Government on lead produced in Canada during 1905 was $334,224. For the fiscal year ended June 30, 1905, the total paid was $337,216, and for the year ended June 30, 1904, $195,284. The bounty decreases as lead advances, and ceases when the London price is £16 or higher per ton, which was the case from Nov. 29 on through the remainder of 1905.

Just after the end of 1905, the smelters of British Columbia reduced the rate for the treatment of silver-lead ore. Previously this ore was smelted at a combined freight and treatment rate of $15 per 2000 ℔., 90 per cent. of the lead being paid for at the London quotation for soft Spanish, less 1c. per ℔. to cover freight, refining and marketing charges. The rate has now been reduced to $12 per 2000 pounds.

This reduction appears to have been due not so much to benefit the miners as to meet the competition of European smelters. For a long while there has been in Europe a scarcity of high-grade galena ore, such as is produced in British Columbia, and European smelters have been offering liberal terms for that class of ore. It was, however, difficult for them to do business in British Columbia so long as the Dominion bounty applied only to lead smelted in the Province. The smallness of the output of many of the mines, and their inability to enter into contracts for regular delivery, also were deterrent to an export business. A couple of years ago, however, the St. Eugene, which is the largest producer of lead in the Province, secured the application of the bounty to ore that might be exported by it under certain conditions, and enteréd into a contract under which a considerable tonnage of ore was shipped to Europe.

The gradually rising price of lead in 1905 finally carried the metal on the London market to the point where the Dominion bounty ceased and put the European smelters on equal terms with the Provincial, in so far as that matter is concerned. Some inquiries for ore for export were made in the local market, which doubtless were instrumental in bringing about the recent reduction on the part of the local smelters.

Mexico.—The unsatisfactory condition of lead mining in Mexico, which we reported in our last volume, continued through 1905, and there was a large decrease in the production, as indicated by the imports of ore and base bullion into the United States, whither most of the Mexican production comes. The total importation from Mexico into the United States in 1904 was 102,903 short tons; in 1905, it was only 87,583 tons. There is a large demand by the smelters in Mexico for lead ore, and every inducement is offered to producers. However, the mines appear to be becoming less productive in depth, or the lead ore is giving place to mixed sulphides, rather high in zinc.

Germany.—The lead production of this country appears to be practically stationary. The statistics of the various smelters are given as follows by Julius Matton, London, in his annual circular:

Company.	1902	1903	1904	1905
Mechernich	19,655	14,654	10,459	11,250
Stolberg-Westphalia	18,339	16,089	16,570	16,466
Rhein-Nassau	11,945	11,908	12,101	11,650
A. Poensgen & Sons	10,550	9,851	10,372	12,235
Ems	4,859	4,259	3,787	4,258
Braubach	18,073	20,147	20,290	20,894
Giesche's Erben	5,869	6,719	8,367	8,945
Tarnowitz	24,004	35,005	31,017	41,610
Harz	14,658	15,269	15,463	13,909
Freiberg	7,931	7,364	5,452	5,146
Anhalt	2,871	2,417	1,661	2,400
Others	....	308	372	301
Totals	138,754	143,990	135,911	148,634

The imports and exports of lead and lead ore to and from Germany during the last three years have been as follows (in metric tons):

Year.	Metal.			Ore.		
	1903	1904	1905	1903	1904	1905
Imports	52,440	61,388	78,528	67,573	83,807	92,667
Exports	30,243	23,169	32,515	1,270	1,312	1,496

Italy.—The lead deposits discovered recently in the Rio Vigneria group of iron mines, Elba, are to be exploited actively by the Société Elba. The production of lead ore in Sardinia in 1905 was 40,000 metric tons, averaging about 60 per cent. lead.

South Africa.—Lead mining still proceeds slowly at the Edendale mine, between Pretoria and the Premier, but the output is small. There are said to be large deposits of galena in the western Transvaal.

Spain.—Judging from the exports, the lead production of Spain in 1905 appears to have been stationary. We have comparatively little information as to the present conditions of Spanish lead production. The following notes on the Linares district, one of the most important of the Kingdom, by Norman Carmichael, are consequently of value[1]:

This district is situated about 200 miles south of Madrid. The mines are situated in an area three miles long and two miles wide. The veins occur in granite, which is covered by soft red sandstone, varying from a few feet to 50 ft. or more in thickness.

The ore is found in true fissure veins, which stand about vertical. The veins may usually be followed for 4000 to 5000 ft. along their strike; they split into numerous small stringers at their extremities. The ore is a low-grade galena, carrying from 4 to 15 oz. of silver per ton, while minor quantities of carbonates are found in the more oxidized portions of the veins.

The ore occurs usually in shoots which may be from a few hundred up to 1500 ft. long, with barren areas of sometimes hundreds of feet between. The veins continue down to a depth of about 2000 ft., near which level all the veins in the district become impoverished. The ore shoots vary in width, but rarely reach more than 5 or 6 ft. The ore is found as small bunches and kidneys in a filling consisting mostly of calcite and kaolinized matter, from the decomposition of the wall rock. The associated minerals, which, however, occur only in insignificant quantities, are copper and iron pyrites, lead sulphate and phosphate and zinc blende. The walls of the veins show very little evidence of movement, and the veins are rarely faulted.

The Quinientes mine of the Linares Lead Mining Company works upon

[1]*British Columbia Mining Record*, August, 1905.

a vein which is opened for 3000 ft. along its strike, and to a depth of 1470 ft. The workings are reached through three shafts. There are 15 levels, the upper ones being 65 to 80 ft. apart, while the lower are 120 ft. apart. The mine is comparatively dry.

The ore is blocked out by levels and raises, the latter being 200 ft. apart, and development work is always kept well ahead of mining. All stoping is done by contract, usually by two contractors to the block, one of whom works from one side and the other from the other side, toward the center. The men mine the ore by overhand stoping on to the timbers, and only as much is drawn through the chutes as leaves them room to work until the block is finished. The contracts for this work can usually be let for about 15 pesetas per sq. m., which is equivalent to about 20c. per sq. ft. The miners are required to take out all of the vein between the walls, so if the vein be 5 ft. wide the cost will be, roughly, 40c. per ton of ore. It is found profitable to mine the vein even when it contains an average of only 1 in. of low-grade galena.

When stoping is finished, the ore is drawn as required; when the stope is exhausted it is filled with waste from development work, if available. If not, stulls are put in to support the walls where necessary, but as the ground is good and stands well, little of this is needed. This is a great advantage, as timber is scarce, and has to be imported at a cost of $30 to $60 per thousand feet. Sometimes underhand stoping is resorted to in places where the ground is particularly good, and when greater tonnage is needed.

All mucking for the mine is done under one contract at about 15c. a skip-load hoisted, the company paying the hoistman.

Shaft-sinking is done by contract at a cost of about $13 to $14 per ft. Winzes cost $3 to $4 per ft. Mucking is not included in the contract for driving.

The contract work has been found the most satisfactory method of working. When work is done by the day the standard rate of wages is as follows: Machine men, $0.75; hand miners, $0.52; hand miners in shafts, $0.60; muckers, $0.52; timbermen, $0.60; timbermen's helpers, $0.52; hoistmen, $0.52 to $0.60; blacksmiths and carpenters, $0.60; laborers, $0.37; women and boys (surface), $0.11 to $0.19.

The process of ore dressing is very simple, as the galena is practically unassociated with other heavy minerals, and not intimately mixed with the gangue matter or disseminated through it in fine crystals. Cobbing, hand-jigging and buddling are the essential features of the process. The mine produces about 200 to 250 tons of ore per day, which is concentrated 10:1 to 15:1, the concentrates assaying about 80 per cent. lead.

The Tortilla smelter, owned by T. Sopworth & Co., is the most important in the Linares district. It employs about 400 men. The smelting is

done by the Scotch hearth method. There are 20 hearths in operation, which have a combined capacity of about 60 tons of bullion per day. Each hearth requires two men who work in shifts of four hours per day, consequently there are 12 men to each hearth per 24 hours. The bullion is run into bars of a convenient size, and then passes to the desilverizing plant, which has 12 Pattinson kettles and the customary cupelling and melting furnaces. The raw bullion contains 12 oz. of silver; this is raised to 500 oz. before the bullion is cupelled, the resulting lead containing less than 0.5 oz. silver. The lead is run into 50-kg. pigs. The Luce-Rozan process is also used here, a two-unit plant served by one steam crane being in operation. The slags from the hearth, which contain about 55 per cent. lead, are resmelted in a blast furnace with iron and lime flux and coke fuel. This furnace is built of sandstone, and its internal dimensions are 47x35 in. It has three tuyeres at each side, and lasts usually about 40 to 50 days.

The galena deposits of Mazaron have been described by Richard Pilz, *Zeit. f. prakt. Geol.*, Nov. 1905, pp. 385–409.

The Lead Markets in 1905.

New York.—In a line with nearly all other staple metals, lead was strong during 1905. The market remains dominated by the American Smelting and Refining Company.

Consumption throughout the year was extremely heavy. There was a heavy drain on the light stocks in existence, which exhausted them completely, with the result that toward the end of the year fancy prices had to be paid by belated manufacturers and consumers, who had not covered their requirements in due time.

On former occasions, efforts were made to prevent the importation of foreign lead whenever market conditions reached a point where such transactions might be profitable. In 1905, however, importations were welcome, as otherwise the scarcity here would have been such as seriously to hamper the trade, and as long as home producers could market every pound of what they were able to produce, there was no longer any necessity of shutting out supplies from other quarters.

At the beginning of the year the American Smelting and Refining Company's prices for refined lead were 4.60c. New York, and 4.52½c. St. Louis, but, more or less unexpectedly, these quotations were reduced, on January 23, to 4.45c. and 4.37½c., respectively. With very slight variations, these figures remained for several months, and it was not until the end of July that the opening prices of the year were again established. From then, prices advanced rather quickly without putting any damper on the demand, which on the contrary became stronger and stronger as the year advanced, at the close of which they were 5.60c. New York, and 5.52½c. St.Louis.

AVERAGE MONTHLY PRICE OF LEAD PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*
1896.....	3.08	3.19	3.14	3.07	3.03	3.03	2.96	2.73	2.77	2.80	4.35	3.04	2.98
1897.....	3.04	3.28	3.41	3.32	3.26	3.33	3.72	3.84	4.30	4.00	2.96	3.70	3.58
1898.....	3.65	3.71	3.72	3.63	3.64	3.82	3.95	4.00	3.99	3.78	3.76	3.76	3.78
1899.....	4.18	4.49	4.37	4.31	4.44	4.43	4.52	4.57	4.58	4.58	3.70	4.64	4.47
1900.....	4.68	4.68	4.68	4.68	4.18	3.90	4.03	4.25	4.35	4.35	4.58	4.35	4.37
1901.....	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.15	4.33
1902.....	4.00	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.069
1903.....	4.075	4.075	4.442	4.567	4.325	4.210	4.075	4.075	4.243	4.375	4.218	4.162	4.237
1904.....	4.347	4.375	4.475	4.475	4.423	4.196	4.192	4.111	4.200	4.200	4.200	4.600	4.309
1905.....	4.552	4.450	4.470	4.500	4.500	4.500	4.524	4.665	4.850	4.850	5.200	5.422	4.707

London.—The year opened with abundant indications of the improving state of trade, there being an active demand from consumers who had previously held aloof. Prices rose to £13 for foreign, and £13 2s. 6d. for English; but the pressure of unsold parcels arriving throughout the month caused prices to drop to £12 15s. and £12 17s. 6d. respectively, notwithstanding the general refusal of producers to make any concessions.

February opened with less offering and with improved demand both for prompt and forward deliveries. The persistent pressure of arriving parcels gradually forced the price down to £12 5s., although buyers were willing to pay premium for forward delivery. The situation was sufficiently clear to bring about a gradual improvement to £12 7s. 6d. for foreign, and £12 11s. 3d. for English; at which figures the month closed rather quietly.

March saw the market depressed by eager sellers at a time when consumers were very reserved, the result being a fall to £11 18s. 9d. for prompt delivery. This, however, was obviously unwarranted by the industrial outlook, and the second week saw a sharp advance to £12 5s. for prompt and £12 10s. for forward. Thereafter manufacturers became busier and had to replenish depleted stocks. The month closed with foreign at £12 11s. 3d. and English at £12 13s. 9d.

April found the improved conditions well established, and further stimulated by increasing demand for export, especially to Russia. The home trade, however, was less active, which checked the advance, and the closing prices were £12 12s. 6d. for foreign and £12 13s. 9d. for English.

May made it increasingly evident that supplies were falling short of trade requirements, and the month closed at £12 18s. 9d. for foreign and £13 10s. 3d. for English.

June was uneventful, but with a gradual advance, due to the steady demand, chiefly for home trade. The closing prices were £13 5s. for foreign and £13 8s. 9d. for English.

July found increasing demand for export, chiefly to Russia and Japan, while the general requirements were such that in many cases a substantial premium was obtainable for prompt delivery. Prices rose steadily throughout the month, closing at £13 15s. for foreign and £14 5s. for English, this

wide difference being attributable to the willingness of bears to sell for forward delivery at the enhanced prices.

August opened with a temporary setback, due to the persistent pressure to sell for forward delivery. Consumers, having covered immediate requirements, were satisfied to let the market drift awhile. But by the middle of the month, a large Continental demand arose. At the close, £14 7s. 6d. was obtainable for foreign and £14 12s. 6d. for English.

September at the outset was marked by an excellent demand for the home trade. When this was satisfied, prices fell in sympathy with other metals, until £13 15s. was accepted for foreign. Toward the close, however, an improvement set in. After touching £14 5s., foreign fell back to £14, with English at £14 5s.

October brought forth an active demand, resulting in an advance to £14 10s. By the middle of the month a further advance of 5s. was established. Thereafter the speculative element forced the price from £15 to £14 12s. 6d.; but the demand again preponderated and the month closed with foreign at £14 17s. 6d. for extended delivery and £15 2s. 6d. for English.

November showed no abatement in the demand. Early in the month £15 5s. was paid for spot.

At the close foreign was held for £16 2s. 6d. to £16 5s., according to time of delivery, and English for £16 5s. to £16 7s. 6d.

December revealed, at the outset, an exceedingly small supply of the metal, and buyers numerous and eager, both for near and distant deliveries. The scarcity of supplies was particularly apparent in the United States, and was such that the American market had to come to Europe for supplies —a reversal of the usual order of things. Home manufacturers, while complaining of slack trade, were alert to cover their requirements—near and distant—at successive advances in price. Before the middle of the month £17 8s. 9d. had been paid, at which point consumers seemed to have satisfied their immediate wants, The price relapsed rather suddenly to £16 17s. 6d., but quickly recovered to £17 2s. 6d., at which good quantities were taken for delivery extending into April. Further improvement developed during the closing days. The closing figures were £17 10s. for foreign, and £17 10s. to £17 12s. 6d. for English lead.

White Lead, Red Lead, Litharge and Orange Mineral.

The production of lead pigments remains a vigorous industry, though it showed but little change from the output of the previous year. The total output of white lead was 122,398 short tons, of which 32,964 tons was marketed dry and the remainder in oil; the combined output in 1904 was 126,336 tons. In 1905, the production of red lead was 16,269 short tons as compared with 13,938 tons in 1904; of litharge, 12,643 tons as against 12,487 tons; and of orange mineral 1000 tons, as against 1125 tons in 1904.

In addition to these carbonate and oxide products, there was an output of 7200 tons of "zinc-lead," a pigment obtained by the smelting of Colorado ores, and 6977 tons of "sublimed white lead," a mixture of lead oxide and sulphate obtained by the volatilization of galena ore.

PRODUCTION OF LEAD PIGMENTS IN THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Short Tons	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons	Value.
1897	7,798	$744,709	$105,804	$9,522,360	8,591	$773,190	477	$76,320
1898	9,160	916,000	93,172	9,391,738	7,460	710,192	541	108,200
1899	10,199	1,070,895	103,466	10,812,197	10,020	1,032,060	928	139,200
1900	10,098	1,050,192	96,408	9,910,742	10,462	1,067,124	825	100,650
1901	13,103	1,448,550	100,787	11,252,653	9.460	979,586	1,087	224,667
1902	11,669	1,262,712	114,658	11,978,172	12,755	1,299,443	867	138,349
1903	12,300	1,385,900	112,700	12,228,024	12,400	1,326,800	1,000	168,000
1904	13.938	1,672,569	(a)126,336	13,896,913	12,487	1,248,691	1,125	168,681
1905	16,269	1,919,767	(a)122,398	12,068,443	12,643	1,422,616	1,000	120,000

(a) The output of "sublimed white lead," a mixed sulphate and oxide of lead, is not included in 1904 and 1905.

IMPORTS OF LEAD PIGMENTS INTO THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1896	1,543,262	$47,450	1,183,538	$52,400	51,050	$1,615	1,359,651	$51,077
1897	1,386,070	46,992	1,101,829	48,988	60,984	1,931	1,486,042	67,549
1898	682,449	25,780	506,739	24,334	56,417	2,021	795,116	37,745
1899	776,197	30,479	583,409	30,212	55,127	3,614	1,141,387	58,142
1900	549,551	25,532	456,872	28,336	77,314	2,852	1,068,793	61,885
1901	485,466	19,369	384,671	21,226	49,306	1,873	977,644	52,409
1902	1,075,839	37,833	506,423	25,320	88,115	2,908	997,494	49,060
1903	1,152,715	40,846	453,284	24,595	42,756	1,464	756,742	36,407
1904	836,077	30,115	587,383	33,788	44,541	1,500	766,469	37,178
1905	704,402	26,553	597,510	34,722	117,759	4,139	628,003	31,106

The white lead business as a whole reflected the general prosperity of the country, in 1905, and was probably greater in volume than that of any previous year in the history of industry. This was aided by the favorable weather conditions during the spring months, when there is always a large amount of painting done, if the weather is such as to permit outdoor work. Extensive building operations incidental to the growth of most of the large cities also constituted a prominent factor throughout the year, and the steadily advancing tendency in the price of pig lead stimulated the demand for all classes of lead products.

The rapid rise in the price of the metal during the latter half of the year led to a total advance of $15 per ton in the price of white lead, the first advance of $5 occurring late in August, the second at the beginning of November, and the last near the close of December. This was wholly disproportionate to the increase in the cost of the metal, which was fully $25 per ton. The effect of the last advance was to check the demand for lead to some extent, especially from the consumers of dry lead, who use it in connection with zinc oxide and barytes in the manufacture of graded leads and mixed paints. These buyers showed a disinclination to accept the new level of prices as permanent, and where they are unprotected by old contracts at the lower figures, purchased rather cautiously, waiting until the stability of existing values should be more thoroughly demonstrated. The year closed,

however, with a good prospect for a large consumption during 1906, based upon the continuance of the conditions which favored the use of paints during the year under review.

There was no increase of corroding capacity during the year, although a large plant was erected by the United Lead Company, and a considerable amount of lead was put down in its beds. It is probable, from present indications, that this lead will he marketed by the National Lead Company, as it is understood that the plant in question will pass under its control before any of the product is ready for the market.

The exclusion of Canadian lead from the United States, by our tariff on the metal, has led the Canadian Government to impose an increased duty (now 30 per cent.) upon the importation of white lead, previously admitted under the merely nominal tax of 5 per cent. Canada imported during the 12 months ending June, 1905, 8688 tons of white and red lead, of which 3240 tons were obtained from the United States.

Inasmuch as British Columbia is a large producer of lead ore, and as there is an important smelting and refining industry in the Province, it was only natural that steps should be taken to develop a manufacturing industry in the Dominion.

The first plant for the manufacture of white lead in the Dominion has been established by the Carter White Lead Company, a well-known American concern, which in this new enterprise had the co-operation of important Canadian interests. A second white-lead plant is to be established at Winnipeg, Manitoba, and at least one other plant is projected.

Manufacture of White Lead.

F. Winteler, in *Zeit. f. angew. Chem.* (1905, p. 1179), described the process for the manufacture of white lead which is practiced at the works of the McDougall White Lead Company, Buffalo, and also at other places.

The lead is melted in cast-iron pots, from which it is conducted by a number of narrow pipes into a brickwork chamber. In this chamber a jet of steam strikes the pipes at an angle of 45 deg., causing the lead to scatter into a fine ''sand.'' The product is screened, and the coarse lumps are sent back to be remelted. A charge of 1500 to 2000 kg. of this lead-sand is introduced into a rotating drum, and 80 kg. of 40 per cent. acetic acid is added in three installments: the first portion on the third day; the second, two days later; and the remainder on the fifth day.

For seven days in all the drums are kept rotating, while a mixture of air, filtered fire gases and a little steam is blown in continually. If the steam is too dry, a little water may be added. If the mass is very dry, the reaction proceeds too slowly. On the other hand, there must not be enough moisture present to form a paste. The waste gases are drawn off by a chimney.

A little unconverted lead balls together in the drum. After seven days the contents are taken out of the drum and are ground, with water, in a pug-mill. From this they run down a wooden trough provided with transverse riffles, similar to those used for gold washing. The grains of uncarbonated lead in the material are thus separated.

Next follows a treatment with soda in settling tanks. Finally the product is either dried, or else, while still moist, ground with oil and prepared for the market.

Recent Improvements in Lead Smelting.

By H. O. Hofman.

Analysis of Refined Lead.—The following is an analysis of one of the leading brands of refined lead[1] made in the United States: Pb, 99.99249; Sb, 0.00127; Bi, 0.00308; Cu, 0.0022; Ag, 0.00020; Zn, 0.00075; Mn, 0.00021; Fe, 0.00178; S, none.

Antimonial Lead.—L. H. E. Lacroix[2] states that the addition of a small amount of sodium to antimonial lead makes it less corrodible, stronger and easier to roll. The alloy can be worked in the lead-pipe press and drawn into wire. In preparing it, antimony is fused in a crucible, lead is added, sodium pushed under the surface of the molten alloy, and the whole stirred vigorously. The mixture commonly used is made up of Pb, 1000; Sb, 15 and Na 1 part. So-called "tempered lead" has been found by E. S. Sperry[3] to consist of Pb, 98.51; Na, 1.30; Sn, 0.08; Sb, 0.11 per cent. The hardness and toughness of the metal is due to the addition of sodium. The small amounts of tin and antimony found go to show that scrap (junk) lead had been used in making the alloy. An alloy manufactured by the Société Routin et Mouraille[4] has a similar composition: Pb, 100; Sb, 1.5; Na, 0.1 part.

Under the heading "Metallics"[5] is a note which states that antimonial lead with 10 per cent. Sb can be rolled satisfactorily, into good sheets without danger of cracking, if the alloy has been cast into the form of a cake at a temperature just above its melting point. Hard lead cast at a red heat cracks badly when it is rolled.

L. P. Pressler, G. B. Putnam, E. R. Richards and E. F. Stoeckly[6] have tested alloys of lead and antimony with from 0 to 20 per cent. Sb, for tension, compression and flexure. The lead used in preparing the alloys was pipe-lead; the antimony had the following composition: Sb, 99.60; As, 0.20; Fe, 0.10; Zn, 0.05; Cu, 0.05 per cent. The results are given in tables and diagrams. The alloy Pb, 94 per cent. and Sb, 6 per cent.

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1 *Brass World*, 1905, I, 267.
2 *Ibid.*, I, 152.
3 *Ibid.*, I, 231.
4 *Metallurgie*, 1905, II, 363.
5 *Engineering and Mining Journal*, 1905, LXXX, 164
6 *Bulletin* Colorado School of Mines, 1905, III, 84.

is the one which shows the highest degree of strength, malleability and toughness combined with a low percentage of antimony.

White Metal Alloys.—G. H. Chandler[1] gives the following formulæ (in pounds) for white metal mixtures which form strong and hard alloys.

Pb.	Sb.	P-Sn.	Sn.	Cu.
65	12		35	2
100	14	1	15	4
100	14	0.25	58	3
31	13		75	3.5

Burning Lead.—M. U. Schoop[2] discusses the burning of lead, and describes two burners, one for large-scale, the other for small-scale work.

Constitution of Lead Sulphides.—K. Friedrich and A. Leroux[3] have investigated the existence of so-called subsulphides of lead. It will be remembered that metallurgists at the beginning of the last century (e. g. Bredberg[4]) held that there existed the subsulphides Pb_2S, and Pb_4S, and perhaps others containing more lead. Percy[5] declared them to be mixtures of PbS and Pb, as he could separate the Pb in excess of PbS by liquation. The authors have finally settled the matter definitely by drawing the freezing-point curve of mixtures containing 0.6–13.5 per cent. S, and examining the different mixtures microscopically. The absence of any maximum in the curve proves the non-existence of a chemical compound, and the horizontal line at about 327 deg. C. represents the solidification of the liquated lead. The nine microphotographs accompanying the paper show only two constituents, viz., galena and lead, which increase and decrease with the proportions of galena and lead in the mixture.

Sulphide of Gold in Galena.—J. N. Nevius[6] calls attention to the occurrence in Sonora, Mexico, of a dark-colored, dull, coarsely crystalline galena with warped cleavage planes, which was found to assay 268.3 oz. Ag and 15.2 oz. Au per ton. Crushing and panning the mineral and examining the heads with the microscope failed to reveal any native gold; the gold must be present in intimate association with lead sulphide, as is more common with silver.

Lead ores in Southwestern Missouri.—C. V. Petraeus and W. G. Waring[7] discuss briefly the character of the lead ores of southwest Missouri, the methods of purchasing the ores and the quality of the lead produced with them. The concentrates shipped to the smelters are very high-grade. Thus, an average of 100 shipments gave 78.4 per cent. Pb.; 14 shipments varying in lead content from 70 to 84.4 per cent. averaged Fe 2.24 per

[1] *Metal Industry*, 1905, III, 48.
[2] *Electrochemical and Metallurgical Industry*, 1905, III, 260.
[3] *Metallurgie*, 1905, II, 536.
[4] Poggendorff's *Annalen der Physik*, 1829, XVII, 268.
[5] "Metallurgy of Lead," London, 1870, p. 39.
[6] *Engineering and Mining Journal*, 1905, LXXX, 769.
[7] *Ibid.*, LXXX, 721.

cent. and Zn 1.78 per cent.; in all of them Bi and As were absent, and only a trace of Sb was found to be present. In the early days the ore was smelted exclusively in the ore-hearth, and this yielded 70 per cent. lead, no account being taken of the lead in the gray slag. Payment was made for the 700 ℔. lead extracted from 1000 ℔. ore and a deduction of $4.50 was made for freighting and smelting. This method of procedure prevails in a modified form today, although other modes of settlement are also found. Thus one buyer will give $3.25 for 1000 ℔. ore whether it assays 75 or 84 per cent. Pb, with pig lead at $4.75 per 100 ℔. at St. Louis; another will pay the same amount for 1000 ℔. of 80 per cent. ore and deduct 50c. per unit for less than 80 per cent. Pb by wet assay; a third will pay 90 per cent. of the lead content at the St. Louis price for pig lead, and make a smelting charge of from $6 to $8 per ton. Pig lead from southwestern Missouri, especially when double refined, to remove small quantities of copper, is sought for glassmaker's litharge and for red lead.

Purchasing Lead Ores.[1]—The rates of the American Smelting and Refining Company for the treatment of gold-silver-copper ores from Clear Creek county, Colo., for 1905 were as follows: Gold, $19 per oz. if 0.05 to 2 oz. per ton; $19.50 per oz. if over 2 oz. per ton. Silver, 95 per cent. of New York quotation, date of assay, when ore assays 2 oz. or over per ton. Copper, $1.25 per unit dry copper ($1.50 off wet) up to 5 per cent. dry; $1.50 per unit dry copper, over 5 to 10 per cent dry; $1.75 per unit dry copper, over 10 per cent. dry; $1 per unit dry copper when lead is paid for. Zinc, 10 per cent. limit, $0.50 penalty for each per cent. over 10 per cent. On ores carrying over 65 per cent. SiO_2, of $14 gross value and under, the treatment charge is $4; $14 to $20 gross value, treatment $5; $20 to $25 gross value, treatment $5.50; $25 to $30 gross value, treatment $6; $30 to $35, $6.50; $35 to $45, $7; $40 to $45, $7.50; $45 to $50, $8; $50 to $75, $9; $75 to $100, $10; $100 and over, gross value, treatment, $11. On ores carrying less than 65 per cent. SiO_2, of $20 gross value and under, treatment, $5; $20 to $25, $5.50; $25 to $30, $6; $30 to $35, $6.50; $35 to $40, $7; $40 to $45, $7.50; $45 to $50, $8; $50 and over gross value, treatment $9. On concentrates of $35 gross value and under, treatment is $3.50; $35 to $80, $4; over $80 gross value, treatment $4.50. Silica limit, 10 per cent., penalty 10 cents per unit excess; zinc limit 5 per cent., penalty 30 cents per unit excess.

Smelting Lead Ores.

Smelting Vanadiferous Ores.—Herrenschmidt[2] describes his process for treating ores containing vanadinite as carried out at the works Bas-Coudray

[1] *Mining Reporter*, 1905, LII, 11.
[2] *Comptes Rendus*, 1904, CXXXIX, 635; *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1905, LIII 201.

at Le Genest, Mayenne, France. The ores coming from the Santa Marta mine in Spain average 12 to 14 per cent. V_2O_5 and 50 per cent. Pb. They are mixed with soda and coal, and fused in a reverbatory furnace, giving as products argentiferous lead and slag consisting of vanadate, aluminate and silicate of sodium. The slag is fused again in a reverberatory furnace in a strongly oxidizing atmosphere, to insure the complete conversion of all vanadium into vanadium pentoxide, granulated and leached. The insoluble residue with 2 per cent. V_2O_5 is treated later with sulphuric acid; the solution, containing 95 per cent. of the vanadium of the original ore, is free from alumina, but retains some silica. In order to separate this, the solution is evaporated to a syrup-like consistency and treated with sulphuric acid of 66 deg. B. The solution with its precipitated silica is now added in correct amounts to an unpurified fresh solution from granulated slag and thoroughly mixed with it, causing a complete precipitation of silica to be separated by a filter pressing. The alkaline filtrate is concentrated, sulphuric acid is added to free the vanadium pentoxide, the solution is evaporated and the excess sulphuric acid driven off. The residue is treated with water, filtered and washed; it assays 92 to 95 per cent. V_2O_5.

Smelting by the Ancients.—F. Freise[1] publishes an interesting review on ''Mines and Mining with the Egyptians, Greeks and Romans'' which forms profitable reading for those interested in the historical side of mining and metallurgy.

Smelting in Greece.—H. F. Collins[2] briefly reviews the present state of ore-treatment at Laurium, Greece. Here only the part relating to lead smelting will be discussed. The materials treated are of two kinds: (1) slag, slime, waste heaps and mine-filling from the ancient Greeks, and (2) ore mined by the French Laurium Company. The average values of the former are: slag, 8 to 13 per cent. Pb, 11 to 17 oz. Ag, per ton; slime, 8 to 10 per cent. Pb, 40 to 50 oz. Ag per ton; waste and mine-filling, 3 to 7 per cent. Pb, 20 to 40 oz. Ag per ton. The waste and mine-filling are enriched by mechanical concentration. The concentrates and slime are briquetted with flue-dust and fine screenings in a soft-mud augur machine with two nozzles 4x6 in. The mixture is piled up to an oblong bed 4 ft. high, holding 60 tons, watered with a hose and transferred to the machine. The prismatic bar is cut into bricks weighing 22 ℔., to be air-dried for two days and then stacked. Slag and briquettes are smelted in Castilian furnaces 9 ft. high with 15 per cent. coke and a blast pressure of 10 in. water. The charge contains 9 per cent. Pb; the base bullion assays 50 to 70 oz. Ag per ton; the slag has the following composition: SiO_2, 25 per cent., Fe (Mn)O, 35 per cent., CaO, 17 per cent., Al_2O_3, 12 per

[2]*Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1905, LIII, 354, 367, 383, 391, 405, 436.
[1]*Engineering and Mining Journal*, 1905, LXXXIX, 363.

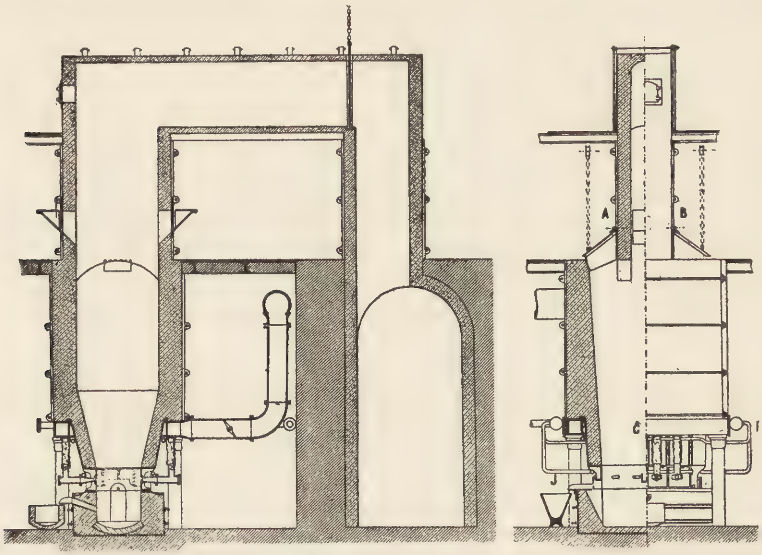
cent., ZnO, 8 per cent., and is fairly free from Pb and Ag. The loss by volatilization is large.

The ore mined by the French Laurium Company is crushed and washed. The concentrates, assaying 50 per cent. Pb, and 20 oz. Ag per ton, are roasted in hand reverberatory furnaces, 39x12 ft., each of which treats in 24 hours four charges of 3300 ℔. with a consumption of 18 to 20 per cent. of poor coal; the sulphur is reduced from 24 to 6-7 per cent. The older blast furnaces are 9 ft. high and 55 in. in diameter, the later ones are 14 ft. high and 60 in. in diameter. The different ores are mixed so as to be self-fluxing; the blast pressure is 7 oz.; the coke consumption is 15 per cent. The subjoined analyses of furnace products show the presence of considerable amounts of arsenic, which in part explains the loss in fume.

	Slag.	Matte.	Speiss.	Base Bullion.	Flue-dust.
SiO_2	25.00	0.80	0.80	....	0.16
Fe	23.63	50.83	63.80	....	....
Mn	3.09	1.66	....	....	...tr....
Al_2O_3	12.00	3.83	4.96	....	....
CaO	16.83	....	....	....	As_2O_3
As	....	0.40	20.75	0.30	31.72
Cu	....	1.85	0.63	0.50	....
Zn	8.00	7.57	1.75	....	3.70
Bi	....	....	....	0.05	....
Sb	....	....	....	0.12	....
S	2.82	21.81	4.61	....	6.43
Pb	0.56	7.75	0.50	98.50	39.45
Ag	0.0006	0.025	0.0109	0.190	0.03

Smelting in Sardinia.—E. Ferraris[1] describes with drawings the silver-lead smelting and the zinc-desilverizing plant of Monteponi, Sardinia. The ores treated are lead carbonate, silicious ore with 10 to 15 per cent. Pb, and lead sulphide. The carbonate ores are screened through a 10-mm. sieve, the oversize goes with the silicious ore direct to the blast furnace, the undersize first to a sintering furnace. There are two hand reverberatory roasting furnaces, 60 ft. long, which turn out in 24 hours about 12 tons of roasted ore, consuming 1.8 tons of English coal or 2.4 tons of Sardinian lignite; and one furnace with fusion chamber to roast matte and to liquate intermediary products. The blast furnace is shown in Fig. 1 and 2. The smelting-zone is encased by cast-steel plates, 1.64 ft. high, and the bosh-walls by boiler-iron plates; both are cooled by water sprays; the steel plates rest on gutter-shaped pieces of sheet iron, which carry off the water; the crucible is enclosed by boiler-plate; the shaft rests upon a cast-iron box, which serves as bustle-pipe and is supported by the usual columns. The chimney has one-third the area of the throat of the furnace, leaving openings at the end which serve for occasional charging and cutting out of wall accretions. The charging openings in common use are at the sides. There are 8 tuyeres with slot-shaped nozzles. The furnace puts

through, in 24 hours, 60 tons charge (50 per cent. ore, 20 per cent. flux, 30 per cent. slag), with 12 per cent. coke and a blast pressure of 2 in. quicksilver. The slag has the following composition: 25 per cent. SiO_2, 33 per cent. FeO, 16 per cent. CaO, 3 per cent. MgO, 2 per cent. BaO, 2.5 per cent. Al_2O_2, 15 per cent. ZnO, small amounts of alkali and S, and 1 to 1.5 per cent. Pb. No matte is formed. The dross from the base bullion contains 90 per cent. Pb, 1.6 per cent. S, 1.4 per cent. Zn, 0.85 per cent. Cu, 0.99 per cent. Fe, and 0.22 per cent. Sb. It is smelted in a reverberatory furnace for lead and matte. The furnace gases



SECTION E F. SECTION G H.

FIG 1 AND 2.—BLAST FURNACE AT MONTEPONI.

pass through an arched wet-condensation chamber, 11.1x21.6 ft., divided into 12 compartments by one longitudinal and five transverse walls, which force the gases to change their direction seven times. There are six atomizers consuming 1.5 liters water per minute; 80 per cent. of the water is vaporized; the remaining 20 per cent. collects in a water-seal bottom. The desilverizing plant contains one softening and one refining furnace, one 14-ton desilverizing kettle with liquating pan and liquated lead kettle, a crucible of 440 lb. capacity for retorting the zinc crusts, and an English cupelling furnace. The dry zinc crust assays 5 to 8 per cent. Ag; the lead liquated from it 0.05 per cent. The market lead is of a high grade: Pb, 99.88; Zn, 0.0021; Fe, 0.0047; Cu, 0.0005; Sb, 0.0030; Bi, 0.0007; Ag, 0.0010. The refined silver has the following composition: Ag, 99.720; Cu, 0.121; Fe, 0.005; Pb, 0.018; Au, 0.003.

[1]*Oesterrichische Zeitschrift für Berg- und Hüttenwesen*, 1905, LIII, 455; *Engineering and Mining Journal*, 1905, LXXX, 781.

Lead Smelting in China.—J. C. Shengle[1] describes the old Tien Pau Shan mines, and the smelting processes carried on there. The ore was a sulphide of lead and copper, rich in silver, which was hand-sorted. An average of the rejected ore showed upon analysis 26.97 per cent. Pb, 20.80 per cent. Cu, 169 oz. Ag. per ton. The method of treatment of the selected ore was heap-roasting, crushing the roasted ore with cylindrical granite rollers, mixing with charcoal and smelting in low furnaces built against a back wall. The resulting matte was heated in a covered trough, the fire being urged by blast and the liquated lead cupelled. The paper contains interesting pictures of the primitive furnaces and many details of the apparatus. A correspondent[2] gives a brief history and description of the smelting plant of the Kwai Yuen mines.

Smelting in the United States.—O. Pufahl[3] publishes notes taken in 1904 while visiting some of the leading lead, silver, copper, zinc and gold plants of this country. Those referring to lead-silver and non-argentiferous lead are given below in abstract.

The Perth Amboy plant in its smelting department has 11 hand reverberatory furnaces, 70x15 ft., and three water-jacket blast furnaces. One furnace is 42x96 in. at the tuyeres and 75x96 in. at the throat; the working hight is 16 ft.; there are 12 tuyeres. The other furnaces are 44x128 in. at the tuyeres and 78x144 in. at the throat; the working hight is 16 ft; there are 16 tuyeres; the blast pressure is 35 oz.; the temperature of the waste gases (60 deg. C.) is recorded continuously. The smaller furnace treats 100 to 120, the larger ones 130 to 140 tons charge in 24 hours, with 12 per cent. coke. The refining department uses the Parkes process. The base bullion is softened in four reverberatory furnaces of 60 and one of 80 tons capacity. A 60-ton furnace has a hearth 17 ft. 9 in. by 12 ft. 6 in., with a metal bath 14 in. deep; the grate is 6 ft. by 2 ft. 6 in. The 80-ton furnace has a hearth 19 ft. 7½ in., a metal bath 18 in. deep and a grate 6 by 3 ft. There are five 60-ton desilverizing kettles, 10 ft. 3 in. in diameter and 39 in. deep. Zinc is worked into the lead by means of the Howard stirring machine; zinc crusts are taken off and freed from adhering lead by means of the Howard alloy press. The desilverized lead is refined in five reverberatory furnaces, four of which have a capacity of 50 tons and one a capacity of 65 tons. The monthly output of refined lead is 5000 to 5500 tons. The zinc crusts are retorted in 18 Faber-du-Faur furnaces fired with oil. A retort is charged with 1200 lb. dry crust and some charcoal. It takes 6 to 7 hours and 50 gal. oil residuum to work a charge, the oil being atomized with compressed air. The retort bullion is cupelled in seven furnaces,

[1] *Engineering and Mining Journal*, 1905, LXXIX, 1035.
[2] *Ibid.*, LXXIX, 1186.
[3] *Zeitschrift für Berg-, Hütten- und Salinenwesen in Preussen*, 1905, LIII, 400; *Engineering and Mining Journal*, 1906, LXXXI, 185, 719, 811, 907, 985, 1034.

the tests of which are lined with magnesia brick. A furnace concentrates 4.5 to 5 tons retort bullion in 24 hours, consuming three tons of coal. The blast is furnished by a steam injector.

The parting department uses the Moebius process. There are 144 cells grouped in 24 divisions. The electrolyte contains per liter 100 c.c. free nitric acid, 17 grams silver and 30 to 40 grams copper; the output of the dynamo is 62 kw.; a cell requires a current of 260 amp. at 1.75 volts; 1 kw. in 24 hours yields 1600 oz. silver. The daily product is about 100,000 oz. silver.

The National plant desilverizes base bullion by the Parkes process. There are four softening furnaces, one of 100, one of 50 and two of 30 tons capacity; one of the 30-ton furnaces is being replaced by one of 120 tons capacity. Softening lasts about 10 hours; the softening dross is smelted in a small reverberatory furnace with pure galena (80 per cent. Pb) from Wisconsin for lead and copper matte (35 per cent. Cu). This is smelted with roasted matte (55 per cent. Cu) in a reverberatory furnace to blister copper, which serves to precipitate silver from sulphate solution in the parting process. The softening-furnace skimmings are desilverized in a small reverberatory furnace with the addition of coal-ashes; the resulting antimony slag is smelted in a small blast furnace (14 ft. high, 8 tuyeres) for hard lead (27 per cent. Sb, 0.5 per cent. Cu, 0.5 per cent. As). The desilverizing kettles stand 200 to 300 charges. The precious metals are removed with two additions of zinc; the first reduces the silver content to 7, the second to 0.2 oz. per ton. The first crust is taken off with the Howard press and retorted, the second is taken with hand skimmers and forms part of the next following first zinc addition. The rich lead, assaying 9.6 per cent. Ag, is discharged from the retort into a kettle lined with firebrick and poured from this into a cupelling furnace through a trough, after having been raised by means of compressed air. The test ring of the cupelling furnace is rectangular and is made up of two sections; it has an oval bottom, round corners and strengthening ribs, and is water-cooled. The test rests on a carriage; the hearth material is marl. Fine silver is cast into ingots weighing 1100 ℔., which go to the parting department. The desilverized lead is refined in reverberatory furnaces holding 28 and 80 to 90 tons lead; the time required to cook a small charge is four to five hours; a large charge requires eight to ten hours. Steam is introduced through four to eight 0.5-in. pipes to aid in the oxidation of the zinc. The refining skimmings, which contain some antimony, are smelted with the desilverized antimony skimmings. The litharge formed is reduced in a reverberatory smelting furnace to a second-class lead. The refined lead is tapped into a market kettle and then molded by means of a Steitz siphon.

The doré silver is parted with sulphuric acid of 60 deg. B. in cast-iron

kettles; the fumes passing off from the kettles are condensed and give acid of 40 deg. B. to be concentrated in evaporating pans. The silver-sulphate solution is drawn off into a settling kettle, where traces of float-gold are recovered. It is then transferred by means of a steam siphon into a precipitating vat, diluted with water and heated with steam, when the silver sulphate is quickly decomposed by means of 125 copper plates 18x8x1 in., suspended by iron rubber-covered hooks. The copper sulphate solution is siphoned off, the silver is transferred to a vat with false bottom, washed sweet, compressed into cakes 10x10x6 in., melted down in a cupelling furnace (heated with oil residuum), refined with niter, the slag being stiffened with brick-dust. The fine silver (0.999) is kept fluid for about 20 minutes under a cover of charcoal, and then poured into cast-iron molds holding 1100 oz., which have been smoked with a coal-oil flame. The residual gold is boiled repeatedly with conc. sulphuric acid, washed, dried and melted down, with some soda, in a graphite crucible. It is 0.995 fine. The copper sulphate resulting from the precipitation of the silver is settled, pumped into a concentrating pan heated with a steam-coil, brought to 40 deg. B. and then crystallized. The crude crystals are dissolved in water until the solution weighs 40 deg. B. and then recrystallized. The second (refined) crystals are broken by rolling, washed with water, dried in a centrifugal machine and separated by sifting into three sizes; two are packed in wooden barrels and shipped, the third (the fines) is redissolved. The mother liquor from the first crystals is concentrated to 60 deg. B. and serves as fresh solvent.

The Helena plant at East Helena, Mont., has 16 hand reverberatory roasting furnaces and 16 Brückner cylinders. The ore is smelted in two blast furnaces having eight tuyeres on a side, and 6-ft. water-jackets. The 4-ton charges are made up on the furnace floor, raised by an electric motor over an incline to the feed-floor and dumped mechanically into the furnaces. The blast pressure is 40 oz. The furnaces have large matte-settling pots from which the waste slag (Pb, 0.5 to 0.8 per cent.; Ag, 0.5 oz. per ton) overflows into slag pots, which are hauled electrically to the dump where the liquid slag is tapped; the skulls are returned to the blast-furnace charges. The matte from the settlers is tapped into flat cast-iron molds, broken to ¼-in. size, roasted in hand reverberatory furnaces and smelted with silicious copper ore for 45 per cent. copper matte. The dust-chambers are partly brick, partly concrete. All machinery is driven electrically.

The Murray plant at Murray, Utah, has 12 hand reverberatory roasting furnaces, 60x16 ft., each of which roasts in 24 hours, with a consumption of 32 to 34 per cent. of Rockspring, Wyo., coal (ash, 5 per cent; H_2O, 3 to 5 per cent) 14 tons ore or 13 tons matte, reducing the sulphur content to 3.5 to 4 per cent. The roasted material is discharged near the fire-

bridge through a tilting door into an iron buggy, which is raised electrically on an incline and discharged into a cooling-bin, where it is sprayed with water. Roasted ore goes to ore-bins, roasted matte to the blast furnaces. Pyritic ores are roasted in 8- to 12-ton charges in 20 Bruckner cylinders, which make one revolution in 40 minutes. A charge is roasted in 24 hours; the sulphur is reduced to 5.5 to 6 per cent.; the coal consumption is two tons. Steam of 40 ℔. pressure is introduced in the cylinder to suck in air; the flue dust formed amounts to 10 to 15 per cent. One man and helper superintend 10 furnaces on an 8-hour shift; one man fires and discharges two furnaces. The gases from the hand roasters and the Brückner cylinders pass into a main flue, 14x14 ft., then through a canal of concrete, 600 ft. long, and ascend with the gases from the blast furnaces (60 deg. C.) the main stack, which is 200 ft. high and 20 ft. in diameter. There are eight blast furnaces, 48x164 in. at the tuyere level, with 10 Eilers tuyeres, 4 in. in diameter, on a side. The working hight is 20 ft.; the feed-floor is 6 ft. above the lower edge of the downcomer; the water-jackets are 6 ft. high; the crucible holds 30 tons of lead; the forehearth is 8x3 ft. and 4 ft. deep; the slag pots have a capacity of 30 cu. ft.; the matte is tapped into flat molds placed upon trucks. A charge weighs four tons and contains 30 per cent. slag shells; the blast pressure is 34 oz.; the coke consumption is 12 per cent. A furnace smelts in 24 hours 166 tons of charge containing 13 to 14 per cent. Pb and 10 per cent. matte; the slag has the following composition: SiO_2, 36; Fe, 23; CaO, 23; Zn, 3.8; Al_2O_3, 4. The base bullion is transferred to 30-ton spherical cast-iron kettles, where it is drossed, sampled and molded by means of a Steitz siphon. The roaster gases drop their coarse dust in brick chambers, from which they pass through pipes of the form of an inverted V, into a concrete flue of 256 sq. ft. cross-sectional area and 2000 ft. length. The silver content of the dust in the brick chamber is 22 oz. per ton; that of the dust in the flue near the stack 8 oz. The dust is briquetted with lime.

The Arkansas Valley plant at Leadville, Colo., has 24 hand reverberatory roasting furnaces, which reduce the sulphur from 20 to 4 per cent., and one Brown horseshoe furnace which leaves 8 per cent. sulphur in the roasted ore. The smelting is carried on in eight blast furnaces, 44x144 in. at the tuyeres and 20 ft. high, having 18 tuyeres. The water-jackets are 8 ft. high. The matte, with 10 to 15 per cent. Cu and 10 to 15 per cent. Pb, is concentrated to 45 per cent. Cu. The dust-flue, one mile long, is of concrete. The flue-dust is briquetted with lime and clayey ores.

The Pueblo plant, at Pueblo, Colo., has 10 hand reverberatory furnaces, 75 ft. long, which have a capacity of 15 tons and reduce the sulphur to 4 per cent.; five Brückner cylinders, which make one revolution in 50 minutes, and treat 24-ton charges in 48 to 60 hours, reducing the sulphur

to 6 per cent.; and two Brown-O'Hara furnaces, each of which handles 25 tons of ore in 24 hours. There are seven blast furnaces, 48x148 in., with 18 tuyeres, which work with a blast pressure of 34 oz.; each puts through 150 tons charge in 24 hours. The furnaces are fed mechanically. A charge weighing four tons is made up in cars on the furnace floor, raised electrically upon an incline to the feed-floor and dumped. A furnace produces 11-15 tons base bullion and matte with 8 to 12 per cent. Pb, which is roasted and smelted with silicious ore for 45 per cent. copper matte. The base bullion is liquated in spherical cast-iron kettles. The dross, squeezed dry by a Howard press, goes back to the blast furnace. The flue-dust is collected in long flues, and amounts to 0.9 per cent. of the weight of the charge. It is mixed with 5 per cent. lime and briquetted with other fine material in a 6-mold White press.

The Eilers plant at Pueblo, Colo., has 15 hand reverberatory furnaces, 60 to 70 ft. long, and 6 blast furnaces, which are hand-fed, each furnace putting through in 24 hours 160 to 180 tons of charge.

The Philadelphia plant, at Pueblo, Colo., treats all the concentrated copper matte produced by the various silver-lead works of the American Smelting and Refining Company in Colorado and Utah.

The Globe plant at Denver, Colo., has 15 hand reverberatory roasting furnaces, of which five are 60x16 ft. and two 80x16 ft. outside dimensions. A furnace treats six charges of 4400 ℔. in 24 hours, reducing the sulphur from 28–30 per cent. to 3–4 per cent. Of the two Brown-O'Hara furnaces with 90 ft. roasting hearth, small slagging-hearth and three external fire-places, one will roast 26 tons ore in 24 hours, reducing sulphur to 3–4 per cent. There are twelve Brückner furnaces. The fireplace has under-grate blast; air is injected into the cylinder by means of steam; a 24-ton charge is roasted in 70 to 90 hours, depending upon the amount of zinc present; a cylinder makes one revolution in 60 minutes. The roasted ore is briquetted with flue dust and lime. The smelting department has seven blast furnaces, 42x144 in., and 18 ft. 6 in. high, with 16 tuyeres; a furnace puts through, in 24 hours, 120 to 150 tons of charge, with a blast pressure of 32 oz. Slag and matte are collected in a double Nesmith pot and conveyed to reverberatory furnaces, 20x12 ft., holding 14 to 15 tons of melted material. After settling one hour, the waste slag (Pb, 0.5 to 0.6 per cent.; Ag, 0.6 to 0.7 oz. per ton) is tapped into 5-ton pots and goes to the dump. The matte (Cu., 10 to 12 per cent.; Pb, 12 to 15 per cent.; Ag, 40 oz.; and Au, 0.05 oz. per ton) is tapped, crushed to pass a ¼-in. ring, roasted in hand reverberatory furnaces and smelted for matte with 45 to 52 per cent. Cu. The base bullion is transferred from the blast furnace to a 33-ton kettle, drossed, poled with steam for about 10 minutes and molded with a Steitz siphon. It contains about 2 per cent. Sb, 200 oz. Ag, and 8 oz. Au per ton. The gases from the blast furnaces pass through

a brick flue 1200 ft. long and then enter the bag-house, which contains 4000 cotton bags, 18 in. diameter and 30 ft. long.

The percentage of blast furnace flue-dust collected during a period of five months in the following smelters is of much interest; Globe 0.5, Pueblo 0.9, Eilers 0.4, East Helena 0.3, Arkansas Valley 0.2, Murray 1.2 per cent., of weight of charge.

The works of the Federal Lead Company near Alton, Ill., treat non-argentiferous galena concentrate, carrying 65 to 70 per cent. Pb. The ore is smelted in 20 water-jacket Scotch hearths with coke as fuel. The furnaces stand in a single row and receive their blast from one main. Branches deliver the air to the chest along the back of a hearth, from which eight 1-in. pipes pass through the back jacket 2 in. above the bath of lead. The crucible of a furnace is 4x2x1 ft., and holds about one ton of lead. The water space in the jackets is only 1 in. wide. Each furnace has a separate hood, communicating with a common flue, 10x10 ft. and 1500 ft. long, through which a fan of 100,000 cu. ft. capacity draws off the gases and forces them to the bag-house, which contains 1500 cotton bags, 25 ft. long and 18 in. in diameter. The collected dust is ignited in the usual way and smelted with gray slag in a water-jacket blast furnace. The 20 hearth-furnaces produce from ore, containing 80 per cent. lead, 200 tons of lead in 24 hours, with a consumption of eight per cent. of Pennsylvania coke; i. e., .70 per cent. of the lead is recovered in the form of bars and 25 per cent. goes into the gray slag. The hearth-lead goes to a 35-ton kettle to be poled with steam for 0.5 to 1 hour, either alone or with liquated blast furnace lead, and then to be molded with a Steitz siphon. It is sold under the brand "Federal" and is 99.9 per cent. pure; the impurities are 0.05 to 0.10 per cent. copper, and traces of nickel and cobalt. The gray slag, sintered bag-house dust, roasted matte, etc., are smelted in a water-jacket blast furnace having 12 tuyeres. The waste slag contains 0.05 per cent. Pb; the blast-furnace lead is liquated in a reverberatory furnace with V-shaped cast-iron bottom; the dross goes back to the blast furnace, the liquated lead to the poling kettle. The matte is roasted in a hand reverberatory furnace 60 ft. long.

Metallurgical Furnaces.—Borchers[1] has brought together illustrations of some of the leading furnaces used in roasting, smelting and refining lead and copper ores, and compiled tables of their dimensions and their work. The tables especially are of great value to the metallurgist, as most of the data are scattered through technical periodicals and are often difficult to locate. While the author has taken evidently all possible care to render authentic the facts collected in the tables, the work would have been more satisfactory to the reader if the sources of information had been added.

[1]*Metallurgie*, 1905, II, 393, 523.

Smelting in the Reverberatory Furnace.—W. R. Ingalls[1] describes the smelting of galena in the reverberatory furnaces at Desloge, Mo., an operation which has been partially stopped. The leading data about the furnace are: hearth 16x11 ft.; grate 8x3 ft.; ratio of grate to hearth area 1:7.3; charges in 24 hours, 3; ore in 24 hours, 10,500 lb; assay of ore, 70 per cent. Pb; gray slag formed, 27 per cent. of charge; assay of gray slag 38 per cent. Pb; men in 24 hours, 6; coal per ton ore, 0.90 ton; yield of lead, 91 per cent., of which 54 per cent. is pig lead from the reverberatory furnace. Cost of furnace excluding chimney, $2000; cost of smelting one ton of ore in the reverberatory furnace, $4.65; cost of smelting gray slag in the blast furnace, $2.35; total cost of treating one ton ore, $7. The paper gives full details in regard to the construction of furnace, mode of operating, etc.

Smelting in the Ore Hearth.—K. W. M. Middleton[2] publishes an extended article on the present practice of smelting lead ore in the ore-hearth in the north of England. The dimensions of hearth found most suitable are: front to back, 21 in.; width 27 in.; depth 8 to 12 in. A hearth 8 in. deep holds 0.75 ton of lead. The side blocks (9 in. thick, 15 in. high, 27 to 28 in. long) are hollow and air cooled; the pipe-stone is water cooled by means of a 2.5-in pipe; the single tuyere passes through the pipe-stone and ends 2 in. above the level of the lead in a slit 12 in. long by 1 in. high; a ledge projecting 1.5 in. gives direction to the blast. The mode of treating ore offers nothing new; two men attend a furnace whether raw or roasted ore is being treated. The fume collected in a dust-chamber is moistened and then smelted in the hearth. It is first fritted together in the cooler part of the charge, near the surface, and is then pushed toward the center. The pot receiving the lead from the hearth has a partition, with opening near the bottom, to permit ladling without necessitating any drossing; it has a fireplace beneath to keep the lead from becoming mushy. When the furnace is stopped, the lead in the hearth is allowed to solidify; it takes two or three hours to melt it again when the furnace is to be started up. A furnace of the above description treats 7.5 tons of 80 per cent. galena in 24 hours and produces 56 to 60 per cent. lead as pig metal. Of ore that has been roasted for two hours, nine tons are treated, with a

Class.	Pb%	Ag %	Sb %	Cu %	Fe %	Total %
Ore lead.........	99.957	0.0200	.....	.....	0.019	99.9960
Fume lead.......	99.957	0.0035	.....	.....	0.019	99.9795
Slag lead.........	99.013	0.0142	0.874	0.024	0.023	99.9482

yield of 65 per cent. lead. Only six tons of raw galena with 65 per cent. Pb can be smelted in 24 hours, giving an output of direct lead of 43 per cent. Ore with 80 per cent. Pb. requires, in 24 hours, 1000 lb. of coal;

[1] *Engineering and Mining Journal*, 1905, LXXX, 1111.
[2] *Ibid.*, 1905, LXXX, 10.

with 65 per cent., this amount rises to 1500 ℔. The analyses (on p. 400) of three brands of Weardale (Durham county) lead are of interest.

Smelting in the Blast Furnace.

Smelting Zinc Retort Residues.—E. M. Johnson[1] publishes notes on some interesting experiments on smelting zinc retort residues at the works of the Cherokee-Lanyon Spelter Company, Gas City, Kan. The residues had the following composition; 10.5 per cent. SiO_2, 38.5 per cent. FeO, 18 per cent. ZnO, 4.8 per cent. S, 2.2 per cent. Pb, 1 per cent. Cu, 10 oz. Ag and 0.03 oz. Au per ton. Missouri galena (1.5 per cent SiO_2, 2.4 per cent. FeO, 9.5 per cent. ZnO, 11 per cent. S, 74 per cent. Pb) was added to the charge to bring up the lead content. The blast furnace was 36x90 in. at the tuyeres. The original hight of 11 ft. was later raised to 14 ft. The water-jackets were 4.5 ft. high and 1.5 ft. wide; the side jackets had a 10-in. and the end jackets a 6-in. bosh. The distance from top of crucible to center of tuyeres was 11.5 in. There were five 3-in. tuyeres on either side. The blast was furnished by a No. 4½ Connersville blower. An average analysis of the product of seven months gave: Base bullion, 87.5 oz. Ag and 1.49 oz. Au per ton; matte 21.6 oz. Ag and 0.06 oz. Au per ton, 7.8 per cent. Pb, 3 per cent. Cu; slag, 31.1 per cent. SiO_2, 37.5 per cent. FeO, 1.5 per cent. MnO, 14.1 per cent. CaO, 10 per cent. ZnO, 0.77 per cent. Pb, 1.26 oz. Ag per ton. The record of the blast furnace during the period was as follows:

Month.	Blast oz.	Tons per day.	Charge.					Extraction.		
			% Pb	% Coke	% Slag	% S	Matte produced.	% Ag	% Au	% Pb
February	21	52.5	9.0	12.0	30.0	3.7	8.0	84.4	83.0	90.3
March	21	44.8	9.7	13.5	37.0	4.0	9.0			
April	21	43.7	9.0	13.5	35.0	4.3	10.0	97.9	70.5	96.6
May	21	49.4	10.0	13.5	30.0	3.5	6.5	95.6	109.5	88.8
July	17	41	9.8	12.5	34.0	3.8	6.0	97.9	90.0	92.9
August	18	47.0	9.3	13.0	32.0	3.7	6.3	86.2	107.5	87.6
September (*a*)	15	51.	7.3	13.0	30.0	2.8	4.6	92.9	94.0	95.6
Average		45.6	9.1	13.0	32.6	3.7	7.2	90.8	92.4	92.0

(*a*) Less residue on furnace than in other months.

It was found that the fire crept up and the yield in metal fell when the residue contained semi-anthracite. Feeding the residue with the coarse components of the charge proved to be the most satisfactory way.

Roasting in General.—A. W. Warwick[2] discusses in a general way the theory and practice of roasting. Developing the change of FeS_2 in an oxidizing roast to Fe_2O_3 without the formation of Fe_3O_4 (viz., $FeS+O_3 = FeO+SO_2$ and $3\ FeO+O=Fe_3O_4$), he calls attention to the frequent presence of Fe_3O_4 in roasted ore, which, while immaterial in smelting ore, is harmful in the chloridizing roasting of silver ore, as it reduces AgCl.

[1]*Western Chemist and Metallurgist*, 1905, I, 179; *Engineering and Mining Journal*, 1906, LXXXI, 318.
[2]*Mining Magazine*, 1905, XII, 196.

He attributes its formation to a lack of air and to $FeS+10\ Fe_2O_3=7\ Fe_3O_4+SO_2$. Calling attention to the dissimilar behavior of pyrite from different sources, he considers first the time necessary to obtain a certain result. Thus, of two Brückner cylinders, 6x12 ft. and 8x22 ft., treating the same kind of ore, the smaller one roasted its charge in 14 hours, while the larger one required 32 to 34 hours, although the ratio of surface area and tonnage was the same. The greater mass in the larger furnace would seem to play an important part in the elimination of sulphur. Observations upon the roasting of two ores in Brückner furnaces gave the following figures:

Time.	Ore: S, 15.2 %; As, 9.7 %; Fe, 22.2 %; SiO_2, 49.8%; diff. 3.1.		Time.	Ore, S 35.06%; As, 1.24%; Fe, 29.40%; SiO_2, 31.35%; diff. 2.95.		Lb. S expelled per hour in 100 lb. pyrite.
	°C	% S.		°C	% S.	
Began to burn	300	15.2	Began to burn	310	35.06	......
1 hour later	510	13.2	10 hours later	530	18.50	1.76
2 " "	....	10.1	20 " "	545	10.00	1.827
3 " "	....	7.4	36 " "	600	3.20	0.403
4 " "	610	5.6	48 " "	775	0.60	0.180
8 " "	675	2.1	....	....	......	......
10 " "	750	0.11	....	....	......	......

The rate of expulsion of sulphur is seen to decrease as the roast progresses. Warwick believes that, in order to do effective roasting in a reverberatory furnace, the effluent gases ought not to contain more than two volumes of sulphur dioxide, and he argues that the Brückner furnace, in which the condition of atmosphere changes from being overcharged with sulphurous gases at the start to having hardly any toward the end, cannot do as satisfactory work as one of the numerous mechanical reverberatory furnaces with stationary hearth, having a uniformly thin layer of ore. These deductions become more pronounced if roasting is looked at from the thermal point of view.

Hand Reverberatory Roasting Furnace.—K. Löwe[1] constructed an automatic charging device for the 14 hand reverberatory roasting furnaces at Pribram, Bohemia, which has proved to be satisfactory. A furnace receives six charges in 24 hours, and each charging used to take about 10 minutes, during which time much smoke and dust passed off into the building. With the new device, charging becomes automatic. The charging opening, 10.23x7.87 in., is lined with a cast-iron frame; in this is placed the charging device, consisting of a steeply inclined fixed steel plate (0.1181 in. thick) and a movable plate suspended by trunnions in such a way that, when the hopper is empty, the movable plate closes the space between the cast-iron lining and the fixed plate, and when it is being filled, the plate recedes and allows the charge to drop on to the hearth.

Lime Roasting of Galena.—This term, suggested by Ingalls[2], simplifies

[1] *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1905, LIII, 4.
[2] *Engineering and Mining Journal*, 1905, LXXX, 402.

greatly the discussion of the Huntington-Heberlein, Carmichael-Bradford, and the Savelsberg processes. In last year's review, it was stated that the general belief was that the sulphur trioxide of calcium sulphate was the leading oxidizing agent in lime roasting. The experiments of Doeltz[1], carried on in a current of carbon dioxide, now show that the endothermic reaction $PbS+CaSO_4=PbSO_4+CaS$ does not take place; on the conrary, there is a tendency for the mixture to react in the opposite direction, $PbSO_4+CaS=PbS+CaSO_4$.

The paper by Borchers[2] was discussed last year.[3] Hutchings[4] recalls the phenomenon noticed by Percy,[5] and observed by himself, that when burnt lime is added to stiffen up the charge in the English reverberatory smelting furnace, the charge is seen to glow for a time. In roasting a sample of Broken Hill galena concentrate, containing 58 per cent. lead, 3.6 per cent. iron, 14.6 per cent. sulphur, and 3 per cent. silica, in a muffle furnace, alone and with the addition of 10 per cent. of pure lime, he found that the lime-bearing charge ignited all over before the pure galena showed any change whatever, and that it heated up considerably above the surrounding temperature, at the same time increasing in bulk. At first no sulphur dioxide was given off, but it was set free later on. The fully roasted lime-bearing charge was found to contain 23 per cent. calcium sulphate, 20.2 percent. being capable of solution in boiling water. When, instead of pure lime, pure calcium carbonate was used, the effect upon the roast appeared to be the same, except that it took a longer time to start the oxidation. Hutchings does not believe that the carbon dioxide has to be first driven off in order to form lime, but holds that, the oxidation once begun, the sulphur trioxide formed decomposes the carbonate, and the heat liberated by oxidation suffices to carry on the process to the end. Experiments with calcium sulphate show that the galena roasted more energetically when dehydrated gypsum was present than when it was absent. Inasmuch as the experiments of Doeltz, noted above, prove that calcium sulphate does not act chemically upon lead sulphide, the effect of calcium sulphate can only be physical, at least during the time that the charge does not sinter. Roasting the above concentrate with 20 per cent. calcium carbonate (11.2 per cent. lime) so as to preclude any fusion gave 0.02 per cent. sulphur as sulphide, 9.3 per cent. sulphur, as sulphate or 9.32 per cent. total sulphur. Operating in the same way with 27.2 per cent. calcium sulphate (11 per cent. lime) gave 0.05 per cent. sulphur as sulphide, and 11.28 per cent. sulphur as sulphate, or a total of 11.33 per cent. sulphur. Mixing the first ore thus roasted with silica and heating for three hours to fritting gave when

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[1] *Metallurgie*, 1905, II 460; *Engineering and Mining Journal* 1906, LXXXI, 175.
[2] *Metallurgie*, 1905, II, 1; *Engineering and Mining Journal*, 1905, LXXX, 398.
[3] *Mineral Industry*, 1904, XIII, 292.
[4] *Engineering and Mining Journal*, 1905, LXXX, 726.
[5] Hofman, "Lead," p. 238.

cool a hard, strong mass, with 6.75 per cent. total sulphur. Hutchings states that, in the Huntington-Heberlein process, the ore when charged into the furnace must still retain 6 to 8 per cent. sulphur, if the converting is to be successful. In such a charge, the energetic oxidation of the sulphide goes on locally, with calcium sulphate as a carrier. When the temperature has risen sufficiently high to cause fluxing, the sulphur trioxide set free by the decomposition of calcium sulphate acts with the air very energetically upon the adjacent sulphides; these in their turn will be oxidized and the process will progress and spread gradually, the whole content of a well-worked converter never being very hot, but only locally in the region in which the oxidation and slagging are going on from below upward.

In view of the great interest awakened by the new processes of lime roasting, Guillemain[1] reviews the theory of roasting lead ores as a whole. He begins with the statement that in roasting a galena rich in silica in a long hand reverberatory furnace, the reaction $PbS + 3O = PbO + SO_2$ is very subordinate, and that the oxide, if it be formed at all, is instantaneously converted into sulphate, according to the reaction $PbO + SO_2 + O = PbSO_4$, the silica either acting as a catalyzer, or separating the grains of galena sufficiently for air to have free access to them. He bases his theory upon the observation that samples from the cooler part of the furnace all show approximately the same percentage of sulphur, i.e., practically no sulphur has been eliminated there.

The sulphate is decomposed only in the hotter parts of the furnace, either by $3\ PbSO_4 + PbS = 4\ PbO + 4SO_2$ or by $PbSO_4 + SiO_2 = PbSiO_3 + SO_3$, as the gases set free in these parts contain enough sulphur to account for all the sulphur in the original ore. According to this theory, the aim in roasting ought to be simply to convert galena into a mixture of $3\ PbSO_4 : PbS$, when application of heat would complete the desulphurization; and the sole object of having a long hearth with its inherent expensive cost of manual labor would be to save fuel. In roasting ore rich in galena and low in silica, a long hearth is of more importance, as in a short furnace the necessary ratio of lead sulphate and lead sulphide cannot be obtained before the temperature becomes too high, causing fritting of the charge. The poorer the galena, the longer the hearth and the more essential a repeated rabbling in the cooler part in order to obtain the required amount of sulphate.

Turning to the Tarnowitz method of smelting lead ores in the reverberatory furnace[2], he passes over or discards the reaction $PbS + 2\ PbO = 3\ Pb + SO_2$ and confines the reducing effect of sulphur to the endothermic reaction $PbS + PbSO_4 = 2\ Pb + 2\ SO_2$. As the furnace is short (16 ft.

[1] *Metallurgie*, 1905, II, 433; *Engineering and Mining Journal*, 1906, LXXXI, 470.
[2] Hofman, "Lead," 1899, p. 105.

2 in.), it is necessary to rabble continuously and to add lime if the necessary amount of lead sulphate is to be obtained before the charge sinters while raising the temperature. He believes that the main effect of the lime is to cool and loosen up the charge, making it porous; incidentally, it may act as a catalyzer; perhaps small amounts of calcium plumbate may be formed,[1] but surely no CaO_2. That the lime need not have any chemical effect upon the charge is proved by the fact that for many years, at the Münsterbuschhütte, Stollberg, blue billy, rolling mill cinder, etc., have been used as a stiffening ingredient.

Guillemain's explanation is that air blown through a galena ore mixture brought to the kindling temperature oxidizes thus: $PbS+3O=PbO+SO_2$ (+98.8 cal.). (Incidentally, at least, the reaction $PbS+4O=PbSO_4$ (+195.4 cal.) ought to take place.)

The superfluous heat generated has to be absorbed, either by $3PbSO_4+PbS=4PbS+4SO_2$(-187 cal.), the lead sulphate having been formed during the blow or in a preliminary sulphatizing roast; or by added substances which may be chemically neutral, or act as catalyzers. The amount of substance added to the galena is regulated by the percentages of total sulphur, by the form in which the sulphur is present, by the quantity of gangue, by the specific heat of the gangue, and by the degree of preliminary heating or roasting. Thus, a rich galena concentrate will require a large amount of addition of high specific heat or a considerable degree of preliminary sulphatizing roasting to keep the temperature within the required limits; even moistening of the hot roasted charge previous to blowing may be necessary. On the other hand, with a low-grade concentrate, the gangue of which is quartz, any addition of foreign material or a preliminary sulphatizing roast may prove fatal to the blowing. The charge may have to be brought first to the kindling temperature by heating in a closed vessel to exclude the air; it may be necessary to add sulphide material (e. g. matte) to obtain the heat necessary for the success of the operation.

In order to reduce the excess heat of charges rich in sulphur, Guillemain suggests the use of gases from boilers, or better still from blast furnaces, when the converter charges may act as filters; he concludes his interesting paper by expressing the hope that the converter in the near future may serve as a smelting furnace instead of being merely a roasting apparatus.

Several papers giving details of the Huntington-Heberlein, the Carmichael-Bradford, and the Savelsberg processes have been published, which supplement, at least to a limited extent, some of the general descriptions that have been available so far.

Thus Biernbaum[2] gives the results he obtained with the Huntington-

[1]*Mineral Industry*, XIII, 292.
[2]*Zeitschrift f. Berg.-, Hütten-, u. Salinenwesen im Preussen*, 1905, LIII, 219; *Engineering and Mining Journal*, 1905, LXXX, 535, 680.

Heberlein process, at Friedrichshütte, which has replaced the hand reverberatory-roasting and the reverberatory-smelting processes. The preliminary sulphatizing roast is now carried on in mechanical circular roasters, six of which are 19.68 ft. and one 26.24 ft. in diameter. A 19-ft. furnace roasts in 24 hours 27 tons of ore; the 26-ft. furnace, 55 tons. In the former practice[1] a reverberatory smelting furnace treated five tons of ore in 24 hours, and a hand-reverberatory roasting furnace, with sinter hearth, eight tons. Making due provision for reserve furnaces, there would be required for an annual capacity of 50,000 tons of ore 15 reverberatory smelting and 15 reverberatory roasting furnaces, or eight Huntington-Heberlein mechanical roasters, 19 ft. in diameter, equal to 4 furnaces 26 ft. in diameter (the latter require 15 h.p.). The comparative data for labor are: One man per day will treat in the reverberatory smelting furnace 0.83 ton of ore; in the hand-reverberatory roasting furnace, 1 ton; in the 19-ft. mechanical roaster, 4.5 tons; in the 26-ft. furnace, 11.8 tons. The figures for fuel tell a similar story: the consumption per ton of ore in the reverberatory smelting furnace is 50.3 per cent,; in the hand roaster, 28.7 per cent.; in the mechanical roaster, 7.3 per cent. The amount of flue-dust of the former practice, 9 per cent., has been reduced to 1.8 per cent. In blowing the charges, it has been found that most of the zinc passes off into the fumes, and does not go into the blast furnace charges, as was formerly the case. This elimination of zinc, the larger percentage of lead, and the lower percentage of sulphur (1.5 per cent.), and the intimate contact of ore and flux in the blown charge, have almost doubled the smelting power of the blast furnaces with a corresponding saving in labor and a diminished (1 per cent.) consumption of coke. The amount of flue-dust recovered in the chamber has been reduced from 1 per cent. of the weight of the charge to 0.64 per cent. It is clear that the new process permits a considerable reduction in the size of the plant. Thus 83,960 sq. ft. required for an annual treatment of 50,000 tons of ore by 15 reverberatory smelting and 15 hand-reverberatory roasting furnaces, are reduced to 11,776 sq. ft.; for four mechanical roasters 26.24 ft. in diameter, and 10 converters of 4 and 10 tons capacity; or the reduction in area is as 7:1. This ratio is further increased by the doubling of the smelting power of the blast furnaces. Lastly, considering the health of the workmen, the cases of lead-poisoning have been reduced from 16.1 to 9.2 per cent. The only disadvantage in the new mode of operating that Biernbaum has found is the difficult and unhealthful character of the work of breaking up the blown converter charges. This he expects to diminish by elevating his new 10-ton converters so that the cakes of blown charge, when dumped, may fall a sufficient hight to be scattered, so that the breaking by hand will he greatly diminished. The concentrated converter gases

[1]Hofman, "Lead," 1899, p. 105.

finally may, at a later date, be utilized for the manufacture of sulphuric acid.

The Carmichael-Bradford process[1], in operation at Broken Hill, N. S. W., consists in feeding a mixture of lead sulphide ore and 10 to 35 per cent. of dehydrated gypsum into a converter on top of a small bed of glowing coal, kept incandescent by a gentle blast, the operation proceeding as in the H.-H. process. Preliminary tests having proved satisfactory, three 5-ton converters were erected, the gases being converted into sulphuric acid in a lead chamber, 100x20x20 ft. A trial run of 108 tons of concentrate (Pb, 54 per cent.; Fe, 1.9 per cent.; Mn, 0.9 per cent.; Zn, 9.4 per cent.; S, 14.6 per cent.; insol., 19.2 per cent.; Ag, 24 oz. per ton) with 27 tons of gypsum, crushed to pass a 1-in. ring, and retained by a 0.25-in. hole (the 5 per cent. fines formed going to the pug-mill) gave a sintered product (Pb, 48.9 per cent.; Fe, 1.80 per cent.; Mn, 0.80 per cent.; Zn, 7.87 per cent.; S, 3.90 per cent.; insol., 21.75 per cent.; Ag, 22 oz. per ton) which weighed 10 per cent. more than the raw concentrate. A second run with 75 tons of concentrate (Pb, 18 per cent.; Zn, 16.6 per cent.; Fe, 6 per cent.; Mn, 2.5 per cent.; Al_2O_3, 3.2 per cent.; CaO, 2.1 per cent.; insol., 38.5 per cent.; Ag, 19.2 oz. per ton) with 22.5 tons of gypsum, gave converted material (Pb,16.1 per cent.; Zn, 14.1 per cent.; Fe, 5.42 per cent.; Mn, 2.25 per cent.; Al_2O_3, 4.10 per cent.; CaO, 8.10 per cent.; insol., 39.80 per cent.; Ag, 17.5 oz. per ton) weighing 11 per cent. more than the raw ore. During this run, 22 tons of sulphuric acid were produced. The cost of desulphurization per ton of ore, with labor at $1.80 per eight hours, gypsum at $2.40 per 2240 ℔., and coal at $8.40 per 2240 ℔., is estimated to be: 0.25 ton gypsum, $0.60; dehydrating and granulating gypsum, $0.48; drying mixture of ore and gypsum, $0.12; converting, $0.24; spalling sintered material, $0.12;0.01 ton coal, $0.08; total, $1.64. Crediting lime in the sintered product, estimated at $0.12, reduces the cost to $1.52. In preparing the ore and gypsum, they are mixed in a pug-mill, then dried, and after coming to a set are ready for the converter. The power plant for a 150-ton daily capacity requires a 50 h.p. engine.

The Savelsberg process is discussed in two papers by Savelsberg[2] and Ingalls[3]. It resembles the Bradford-Carmichael process in starting with raw ore; and differs from it in that it uses crushed limestone instead of dehydrated gypsum as diluent and flux. This process has been in successful operation at the silver-lead works at Ramsbeck, Westphalia, for the last three years. The converter is a bowl-shaped cast-iron vessel 6.56 ft. in diameter; it weighs 2860 ℔., and has a capacity of 8 tons charge. It is supported on a truck by trunnions, and can be listed by hand power, as shown in Fig. 3. On the bottom is grating, B; and beneath it the blast-

[1]*Engineering and Mining Journal*, 1905, 778.
[2]*Mining Magazine*, 1905, XII, 391.
[3]*Engineering and Mining Journal*, 1905, LXXX, 1067.

inlet pipe. The charge is made up of galena concentrate and 15 to 20 per cent. limestone. A typical charge at Ramsbeck consists of 100 parts of galena (60-78 per cent. lead, 15 per cent. sulphur), 10 parts silicious silver ore, 10 parts spathic iron ore, and 19 parts limestone. It averages about 11 per cent. silica, and may contain 7 to 8 per cent. zinc. With Broken Hill concentrate carrying 10 per cent. zinc, or if the blown charge shows powdery parts, the proportions of silica and iron have to be changed to make a more

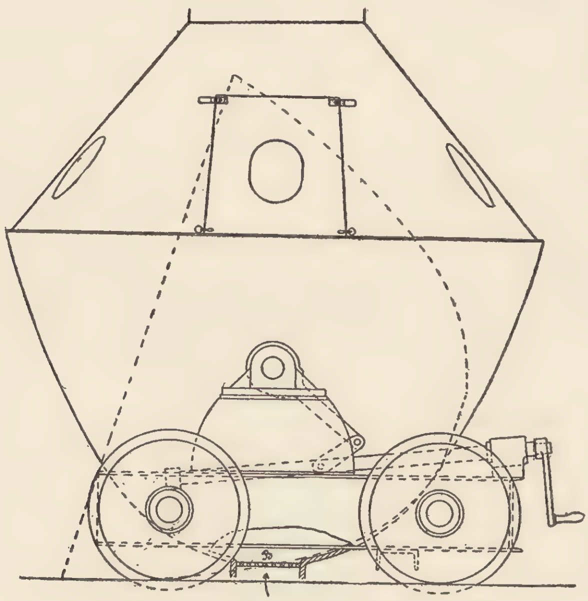

FIG. 3.—THE SAVELSBERG CONVERTER.

fusible charge. Ore and flux are pulverized separately in rolls or ball-mills to pass a 3-mm. screen; they are bedded and mixed twice by raising with a bucket elevator and dropping on the floor; the second cone is flattened out and 5 per cent. water added. The effect of water is to prevent unmixing of ore and flux, to keep down the temperature, and to aid in the formation of sulphuric acid. In starting the converter, the grate is covered with a layer of limestone; this is followed by a layer of glowing coal or coke; and this again is covered with a second layer of limestone. The blast is admitted, and a layer of moistened charge about 12 in. deep introduced. When the surface becomes red hot, a second layer is fed, and so on until the converter is filled. While charging, about 247 cu. ft. of air is forced in at a pressure of 2.75 to 4.5 oz.; the amount is now increased until the pressure reaches 11.5 to 13.5 oz. Desulphurization is followed by scorification; but the charge never becomes liquefied, as the formation of slag is an endothermic reaction and the passage of the air chills any ten-

dency to its becoming fluid. An 8-ton charge is finished in about 18 hours. The converter is tilted, and the cake of blown charge dumped upon a vertical iron bar, and thus broken into several pieces, to be further reduced in size by wedging and sledging. The sulphur of a well blown charge is reduced to 2 or 3 per cent.; one man attends a converter in a 12-hour shift. The labor for mixing and conveying materials, breaking up of product, etc., is extra.

Comparing the three processes, the Huntington-Heberlein requires a preliminary roast and is therefore expensive; the Carmichael-Bradford is confined to gypsum, which is not always available, and requires dehydration; the Savelsberg uses limestone, which is, as a rule, to be had wherever lead ores are mined.

The Huntington-Heberlein process is controlled[1] in this country by the American Smelting and Refining Company, and has been introduced at Pueblo, Colo.; at Murray, Utah; at East Helena, Mont., and at Marysville, B. C.

Bormettes process.—Lotti describes[2] the Bormettes method of lead and copper smelting. The main features of the process are the incorporating of sulphide ore into fluid slag and stirring, whereby a spongy mass is to be produced by an incipient roast, and the blowing of the partly desulphurized product in the converter to eliminate the rest of the sulphur and to cause a partial fusion of the charge. Judging by the success of the different lime-roasting processes, the converting part of the method sounds reasonable; details are missing about the stirring in of dry or slightly moistened sulphide ore into the fluid slag.

Cost of blast furnace.[3]—A recent bid for the iron work of a 36x108-in. water-jacket lead-blast furnace, together with a 10x7-in. rock-breaker, sampling rolls, rotary blower, engines and boilers, power-transmitting machinery, and in fact the smelting plant complete, with the exception of foundation, building and erecting, was $13,000 f. o. b. maker's works. The weight of the machinery and iron work being 170,000 lb., this was an average of 7.65c. per ℔. Another bid for the same equipment was $11,750 f. o. b. works; the total weight in this case being 178,000 ℔., giving an average of 6.6c. per ℔.

First Use of Cast-Iron Water-Jackets in Lead Blast Furnaces.—The accompanying sketch of a cast-iron water-jacket (Fig. 4) prepared by E. Daggett[4] in 1872 for the blast furnaces of the Winnamuck Mining Company, Bingham Canon, Utah. represents, as far as known today, the first cast-iron water-jacket ever used in a lead blast furnace. In a note addressed in 1885 to the *Engineering and Mining Journal*[5], Daggett claimed then

[1] *Engineering and Mining Journal*, 1905, LXXX, 18, 932.
[2] *Ibid.*, 1905, LXXX 580; *Mining Magazine*, 1905, XII, 426; *Metallurgie*, 1905, II, 353.
[3] *Engineering and Mining Journal*, 1905, LXXX, 164.
[4] Private communication, Dec. 23, 1905.
[5] 1885, XL, 318.

that he had been the first man to use a cast-iron jacket in connection with lead smelting, and this claim has not been disproved. Records[1] show that previous to 1872 wrought-iron jackets inclosing the furnace to the feed floor had been used. The following details are of sufficient interest to be placed on record. At the Winnamuck smelter[2], the smelting zone of the blast furnace was usually built of firebrick, which with a slag running as

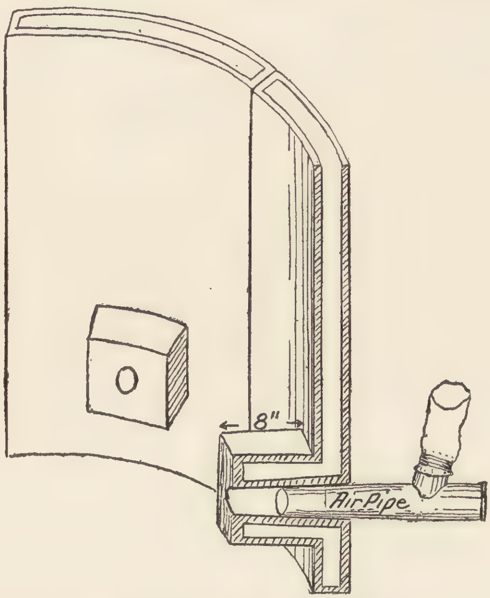

FIG. 4.—CAST-IRON WATER-JACKET.

high as 38 per cent. silica rarely lasted longer than 26 days, the average duration being 16 days. The water-jacket shown in Fig. 4 was cast by Hall & Brothers, foundrymen, Council Bluffs, Ia., and was put in use November 26, 1872. Five or six sections were required in the furnace, which was 3 ft. 6 in. in diameter at the tuyeres. The tuyere nozzle, projecting 8 in. into the furnace, was made square in order that firebrick might be built around it advantageously. The lining, however, was slagged off in a few days, so that about three months later the furnace was started without any lining. In 1873 the protruding water-cooled nozzle was omitted from the new jackets. These had openings large enough to receive water-cooled cast-iron tuyere nozzles, which reached 8 in. into the furnace. This form was used later at the Flagstaff and other furnaces in Utah.

Water-Jackets and Spouts.—H. W. Hixon[3] publishes an interesting article on the evolution of tapping-jackets and spouts. In lead furnaces, the

[1]Hofman, "Lead," 1899, p. 242.

[2]Raymond, "Mineral Resources West of the Rocky Mountains," 1873, p. 360; *Trans.* Am. Inst. Min. Eng., 1873-74, II, 17.

[3]*Engineering and Mining Journal*, 1905, LXXX, 673.

first form of tapping-jacket was and in many cases still is a hollow water-cooled casting 5 in. deep with a conical opening 8 and 2.5 in. in diameter. With furnaces that are not tapped periodically, but from which slag and matte overflow continuously, as is the case in most copper furnaces treating sulphide ores, a water-cooled spout trapping the blast is in common use. At first the trough was made of pipe-coils covered with sheet iron and lined with a quartz-clay mixture. As the stream of slag and matte readily cut a hole into the breast-jacket, the inlet-pipe for the spout was run underneath the jacket. Later, a bronze (or copper) tapping-jacket was placed beneath the breast-jacket and a water-cooled spout of flanged steel bolted to it. The flanged steel was replaced in some instances by a pipe-coil, around which iron had been cast. Copper poured from a converter, serving as a casing for the coil, gave equally good results. The discharge end of the spout was tipped with a water-cooled copper (or bronze) jacket. In smelting nickel-copper ores, the resulting matte has a strong cutting effect (similar to a matte of the same copper content, but having nickel replaced by iron). At the Canadian Copper Company's works the above mentioned iron spout has been replaced by a water-jacketed fore-hearth built of chrome-brick and having a bronze water-cooled overflow nozzle. The best tap for the settler of the furnace is a carbon block, similar to that used for electrodes, 5 in. thick with a bore 2 in. in diameter; the carbon is protected by a cast-iron plate.

Blast-furnace Feeder.—M. Maurer[1] describes, with photographic reproduction, a trolley-car with weighing attachment. The car is run underneath the chutes of the ore and flux bins, secures the desired quantities, and is then hauled to the feed-floor of the furnaces, where it delivers its charge.

Magnetite as a Flux.—J. W. Neill[2] states that at the blast-furnaces of the Montana Ore Purchasing Company as much as six and eight tons magnetite are added in 24 hours to a furnace treating 300 tons charge to help along the smelting with highly calcareous slags. Colquhoun[3] states that magnetite was used by the Arizona Copper Company when treating oxide, but not when treating sulphide ores. Why magnetite should do any harm in the latter case is not clear.

Ore Bedding at the Copper Queen Smelter.[4]—The Copper Queen smelter has a new system of bedding ores and fluxes, and of delivering the mixture to the blast furnace. Ores and fluxes arrive in side-dump cars and are discharged into stone-lined pits in proper amounts to form correct smelting mixtures. These are then loaded by steam-shovel into charge-cars, trains of which are run to the feed-floor of the blast furnaces. The system is doing satisfactory work, the slags formed having the calculated compositions.

[1]*Iron Trade Review*, March 30, 1905.
[2]*Mining and Scientific Press*, 1905, XC, p. 69, 133.
[3]*Ibid.*, p. 69.
[4]*Engineering and Mining Journal*, 1905, LXXX, 197, 309.

With a lead-smelting plant treating a variety of ores arriving in small lots and not a few kinds arriving in large shipments, e. g., cupriferous pyrite and silicious ore, this mechanical bedding and loading is not directly applicable, as the composition of the lead would not be sufficiently uniform. A method gaining in favor with lead smelters is to bed small lots in the accustomed way, but to dump large shipments into elevated bins, make up the charges by running definite amounts into charging-cars resting on platform scales, transport the cars electrically to the furnaces and then discharge them into the feed-openings.

Blast-furnace Slag.—The blast-furnace slag[1] produced at the Port Pirie works of the Broken Hill Proprietary Company has the following average composition: 25 to 26 per cent. SiO_2, 31 per cent. FeO, 5 to 5.5 per cent. MnO, 15.5 to 17 per cent. CaO, 13 per cent. ZnO, 6.5 per cent. Al_2O_3, 3 to 4 per cent. S, 1.2 to 1.5 per cent. Pb (wet assay), and 0.7 oz. Ag per ton.

One of the blast furnaces ran continuously for four years and 10 months. The percentage of ZnO in this slag is high. In the Lower Harz,[2] slags running 20 per cent. ZnO are made regularly; American practice aims to limit the ZnO to 6 per cent.

Copper and Copper Matte in Blast-furnace Slags.—Vallety,[3] in experimenting to determine the amounts of metallic copper and of matte held in suspension by slags, found that in treating the slag with silver nitrate, (1) copper displaced the silver entirely from its solution, (2) copper oxide gave a precipitate of basic nitrate soluble in nitric acid of 5 per cent. HNO_3, (3) silver nitrate and matte acting upon one another formed silver, silver sulphide and cupric nitrate, (4) silver nitrate did not act upon copper silicates.

Fluorine in Blast-furnace Slags.—Two articles by L. S. Austin[4] and E. Kneeland[5] shed much light on this vexed question. The theory that adding fluorspar to a blast-furnace charge assists in volatilizing silica as silicon fluoride has probably been generally abandoned, although it is still found here and there in technical literature. The effect of fluorspar is mechanical; it melts above 1400 deg. C., and when melted is very fluid and thus assists in liquefying slags, which otherwise will not flow readily. As calcium fluoride is not decomposed in the blast furnace, its lime is not available as a base to flux silica. In determining in the usual way the silica of a slag, when calcium fluoride is present, some silica is sure to be volatilized, and the percentage of silica will be too low; in determining the lime, the available portion will be found to be too high, if the percentage of fluoride is not taken into account.

Austin found that by considering the lime, which remained insoluble

[1] *Engineering and Mining Journal*, 1905, LXXX, 404. [2] Hofman, "Lead," 1899, p. 293.
[3] *Engineering and Mining Journal*, 1905, LXXIX, 1223; *Annales de Chimie Analytique*, 1905, X, 193; *Mining Magazine*, 1905, XII, 61.
[4] *Engineering and Mining Journal*, 1905, LXXX, 865, 1222.
[5] *Ibid.*, 1905 LXXX, 1030, 1212.

when the ore was being treated with acetic acid, as being in combination with fluorine, he obtained results in the blast furnace which were satisfactory. Mexican ores, especially those from the northern part, often contain fluorspar. Thus Nina Vera ore contains 15 to 17 per cent. CaO in combination with fluorine, and Veta Grande ore often 10 per cent.

Kneeland combats the statement that has often been made that the presence of fluorspar in a charge causes an imperfect reduction on account of the charge melting too quickly. He found that the amount of coke remained the same, whether his slag contained 1 or 3 per cent. F. He does not consider Austin's method of approximating the percentage of calcium in combination with fluorine reliable, as there are many compounds of calcium which are not decomposed upon being digested with acetic acid. He has used the following method of determining fluorine, with which he has run through four to five assays a day in addition to his regular work: Heat 0.5 to 1.0 gram of ore with 5 to 10 grams potassium-sodium carbonate to quiet fusion; bring to red heat; pour contents upon an iron plate; break crucible into small pieces and place with fused mass in a 6-in. agate casserole. Add 20 c.c. water, digest one hour, near boiling point, remove any undecomposed lumps, grind them in an agate mortar and return to casserole. Boil 10 minutes, filter through loose filter into 1-liter beaker, wash with hot water, then with hot solution of ammonium carbonate, and discard residue. Add to filtrate 10 grams ammonium carbonate, boil five minutes and allow to stand in the cold for two hours. Filter, decanting through loose filter, into 6-in. agate ware casserole, wash once or twice, eliminate silica from filtrate by adding 20 c.c. emulsion of zinc oxide in ammonia, and boil, leaving casserole uncovered until ammonia is expelled. Filter into No. 5 beaker; wash with hot water; add calcium chloride solution to the filtrate, stirring until no more precipitate forms; allow to settle; filter and wash with hot water, testing filtrate for carbonates and fluorine with calcium chloride. Transfer precipitate into platinum dish, dry, ignite at a red heat, cool; then disintegrate in hot water, add acetic acid until clear, evaporate to dryness, avoiding scorching; moisten with acetic acid and evaporate until odor of acetic acid ceases. Wash mass with hot water into No. 3 beaker, add hot water until lime acetate is dissolved, give additional 150 c.c. hot water and stir, keep for a few minutes on a hot plate, filter, wash with hot water, with ammonium chloride, and again with hot water; transfer precipitate to platinum dish, dry, ignite, cool, moisten with water, add 6 c.c. strong sulphuric acid, heat for a few minutes, cool, transfer to No. 2 beaker. Add 5 grams ammonium chloride, boil for a few minutes, cool, add 6 c.c. of strong ammonia, then 2 to 3 c.c. strong hydrogen peroxide, boil and filter. Precipitate lime from filtrate with ammonium oxalate and titrate with potassium permanganate. ($CaO \times 1.392 = CaF_2$. $CaF_2 \times 0.4872 = F$.)

Speiss.—Two papers by Hübner[1] and Huhn[2] on the refining of speiss produced at Freiberg, Saxony and at Oker, Prussia, are of special interest, as these products (of minor importance here) have not received the attention they deserve.

At Freiberg the speiss of the lead blast furnace (4.9 per cent. Ni, 2.4 per cent. Co, 0.07 per cent. Ag, 4.8 per cent. Pb, 7 per cent. Cu) was concentrated as early as 1858 into an enriched speiss (32.5 per cent. Ni+Co, 00.5 per cent. Ag, 0.8 per cent. Pb, 5 per cent. Cu) by fusing with an equal weight of flux consisting of 66 per cent. barite and 34 per cent. quartz. With the enriched speiss there was formed some matte containing copper and nickel; the latter was recovered by smelting with 30 per cent. arsenopyrite and 100 per cent. old slag (50 per cent. SiO_2, 35 per cent. FeO). Preliminary experiments had established the valuable fact that as long as the tenor of iron in the speiss to be refined exceeded 25 per cent., a matte containing silver, lead and copper could be produced which separated readily from the speiss, while this did not take place if the iron content was lower (e. g., 20.5 per cent. Fe, 26.9 per cent. Ni, 9.8 per cent. Co, 0.08 per cent. Ag, 4.9 per cent. Pb (Bi, Sn), 3.6 per cent. Cu), as the sulphide was held in solution by the arsenide. In recent years the composition of the speiss changed considerably. Thus a sample made in 1893 gave upon analysis: 17.8 per cent. Fe, 0.8 per cent. Zn, 11.30 per cent. Ni+Co, 0.6 per cent. Mn, 0.077 per cent. Ag, 11.2 per cent. Pb, 24.3 per cent. Cu, 23.4 per cent. As, 6.5 per cent. Sb (Sn), 0.007 per cent. Pt, 0.001 per cent. Au, 3.5 per cent. S, showing the presence of gold, platinum and antimony. Smelting with galena and matte-slag in the blast furnace collected the gold and part of the silver with some copper in a matte, but the speiss still retained 0.05 per cent. Ag and 20 per cent. Cu; further, its content of nickel could not be made to exceed 14 per cent., and the antimony and platinum not being satisfactorily transferred to the lead, the imperfect extraction of platinum and the high price of this metal were the leading causes for trying new methods. In the first method, the speiss was roasted and treated with sulphuric and hydrochloric acids, but only 70 per cent. of the copper and 65 per cent. of the nickel could be extracted, the probable cause being the insolubility of arsenates of copper and of nickel. Fusion of two samples of speiss (1. Ni, 4.1 per cent.; Ag, 0.10 per cent.; Pb, 6.6 per cent.; Cu, 25.9 per cent.; Sb (Sn), 5.6 per cent.; Pt, 0.004 per cent.; Au, 0.004 per cent. 2 Ni, 20.5 per cent.; Ag, 0.07 per cent.; Pb, 1.7 per cent.; Cu, 16.7 per cent., Sb (Sn), n. d.; Pt, 0.022 per cent.; Au, 0.066 per cent.) with sodium bisulphate in the laboratory followed by leaching was more satisfactory, as it gave a yield of 85 to 97 per cent. of Ni+Cu. Difficulties, however, were encountered in a working test in leaching (platinum going into solution), in precipitating the copper

[1] *Glückauf*, 1905, XLI, 6. [2] *Ibid.*, 1905, XLI, 1165.

(hydrogen sulphide being cumbersome and copper sulphide oxidizing too quickly, scrap iron giving an impure precipitate, electrolysis being too expensive), and also in precipitating the nickel (sodium carbonate gave as best precipitate: Fe, 9 per cent.; Zn, 3 per cent.; Ni, 32.5 per cent.; Co, 2.2 per cent., PbO, 0.2 per cent.; Cu, 3.4 per cent.). Sintering of roasted speiss with sodium nitrate and carbonate, followed by leaching with water (to extract the sodium arsenate and antimonate), and with sulphuric acid (to remove the sulphatized nickel from the oxidized copper) proved only partly successful, as the antimony was not completely soluble, as was the arsenic. The residue fused with a reducing agent gave a reddish button of speiss: Ag, 1.48 per cent.; Sb, 13.6 per cent.; Pt, 0.09 per cent.; Au, 0.14 per cent. The last method experimented with was more satisfactory. The speiss was crushed to pass a screen with 5 mm. openings, roasted dead (sample treated with HNO_3, spec. gr. 1.2, gave only a little brownish fume), mixed with 20 per cent. pyrite of the weight of the raw speiss, and smelted, when heavy white fumes of arsenious and antimonious oxide were given off, the pyrite having a reducing effect upon arsenic and antimonic oxides. But here again antimonic oxide was more difficult to reduce than arsenic oxide. The correct degree of roasting and the necessary additions of pyrite were ascertained by small tests in which weight and color of speiss buttons formed the criterion. At present the speiss of Freiberg, freed from its arsenic as shown, is smelted in the blast furnace with pyrite, galena, litharge low in silver and matte-slag. The products are base bullion, speiss, matte and slag. The speiss (25 per cent. by weight of the arsenical speiss) separates readily from the matte. In the table given herewith, samples from 1897, 1902 and 1903 show little differences in composition, in comparison with the average of 1904, with its very high percentage of antimony.

Year.	Fe.%	Ni.Co.%	Ag.%	Pb.%	Cu.%	Sb.%	Pt.%	Au.%
1897	....	11.7	0.06	6.0	19.4	....	....	0.0008
1902	....	29.0	0.07	4.0	10.0	....	0.080	0.0002
1903	....	31.1	0.05	....	12.1	....	0.020	....
1904	24.8	19.6	0.06	7.1	10.9	27.0	0.014	0.0002

The ores treated at Oker come mainly from the Rammelsberg mine. They are divided into five classes, which in 1902 averaged as follows:

Class.	Au.%	Ag.%	Cu.%	Pb.%
Copper ore No. 1	0.00013	0.017	18.37	3.85
Copper ore No. 2	0.00015	0.012	12.07	1.70
Copper ore No. 3	0.00011	0.008	5.44	1.25
Mixed ore	0.00025	0.018	4.98	8.10
Lead ore	0.00007	0.015	0.37	9.15

They consist mainly of pyrite, chalcoprite, blende, galena, barite and 6 to 10 per cent. gangue, which is principally slate and limestone (or dolomite).

The lead ores contain 0.05 per cent. As; this increases to 0.15—0.20 per cent. with copper ore No. 3, and falls to 0.10—0.15 per cent. with copper ore No. 1. The ores are roasted and smelted in blast furnaces. In concentrating roasted lead matte in the blast furnace there is formed, in addition to some lead, a 35 to 40 per cent. copper matte, and a rich speiss (with about 50 per cent. Cu, 35 per cent. Pb, a little Fe, and 6 per cent. As and Sb). It is "mushy" when hot, and has, when cold, a fine grained fracture, a light-gray color with a reddish tinge, and a sp. gr. of about 8. The poor or common speiss from other operations assays Cu 18 to 27 per cent.; Pb 16 to 20 per cent., and contains much Fe, As, and Sb; it is brittle, silver-white with a reddish tinge, and has a fracture similar to spiegeleisen. The two following analyses are typical:

	% Cu.	% Ag.Au.	% Pb.	% Bi.	% As.	% Sb.	% Fe.	% Co.	% Ni.	% Zn.	% S.	% Total.
Rich	51.73	0.175	35.20	1.63	2.75	3.34	1.65	0.24	0.13	1.82	1.38	100.045
Poor	25.85	0.085	16.68	—	11.04	13.50	22.17	1.11	1.60	3.31	4.13	99.475

The method of working these speisses is to subject them in a reverberatory furnace, with surface blast, to a roast-smelting. The products are copper to be granulated and slag. The copper granules have the following composition: Cu, 91.470 per cent.; Ag, 0.462 per cent.; Au, 0.0154 per cent.; Pb, 0.887 per cent.; Bi, 0.286 per cent.; As, 3.863 per cent.; Sb, 2.133 per cent.; Fe, 0.028 per cent.; Ni+Co, 0.750 per cent.; total 99.8944 per cent.; the slag goes back to the blast furnaces. There are two reverberatory smelting furnaces, the hearths of which are built in iron pans, water-cooled at the sides and bottom, while the roof is movable. One furnace has a hearth 10.33x5.74 ft. and a grate 5.58x1.64 ft.; the hearth of the other is 11.15x6.56 ft. and the grate 5.77x1.64 ft. Both furnaces have under-grate blast. The bottom of the hearth consists of a layer of marl tamped down firmly; it is followed by a course of brick, laid dry. The working bottom rammed into place is a mixture of 20 parts shale and 80 parts marl ground to pass a 1-mm. hole. The metal bath in furnace No. 1 is 0.48 ft. deep; in No. 2, 0.69 ft.; the copper is tapped from both sides of the furnace into granulating basins. The mode of operating is as follows: Charge 3.5 tons poor speiss into the smaller, or five tons into the larger furnace; bring to a red heat and roast; melt down when no more fumes are given off; skim the dross which has a metallic character, and turn on the blast (38 to 40 mm. Hg); skim the slag which takes up iron, antimony, lead and zinc in the order given; lower the temperature when the slag ceases to form, until the bath becomes pasty, and turn off the blast to assist volatilization of arsenic; fire up again when arsenic fumes cease to be evolved· continue cooling and firing up until no more arsenic

can be volatilized. The speiss in the furnace has become enriched to 75 per cent. copper. Charge 2.5—3 tons of rich speiss; fire up; roast as much as possible and liquefy charge; skim the dross; turn on blast; skim slag; cool and re-heat as before to drive off arsenic. In order to obtain satisfactory granules, the copper content ought to range between 90 and 93 per cent. It takes 40 hours and 1.43 tons of coal to refine the 6.5 tons of charge in furnace No. 1; the 1.3 tons of hearth material, forming the working bottom, lasts nine charges. With furnace No. 2, the 8-ton charge takes 48 hours and 1.6 tons of coal; the 1.6 tons of hearth material of the working bottom last six charges.

Wall Accretions.—A note[1] states that instead of cutting out the wall accretions in a lead blast furnace, metallurgists at present charge a small amount of salt cake along the sides of a furnace, which soon removes the hangings.

Flue-dust.—J. C. Hain[2] discusses the disentegration of portland cement briquettes by oil, and the means of preventing it. In view of the fact, discovered by Messiter[3], that dust flues of cement concrete should not be painted with linseed oil, as the cement attacked the oil, the Hain paper has a special interest for the lead smelter. He states that tests, with comparatively new briquettes of neat cement (1:3 of sand and 1:3 of lime stone screenings), weakened by exposure to air, had shown they were readily disintegrated. Thus a sand briquette was disintegrated by signal oil (a mixture of mineral oil and animal oil) in six months to such an extent that it could be crushed by the hand. Of the substances investigated, animal fat had the strongest disintegrating effect; then followed animal oil, vegetable oil and mineral oil. Drying vegetable oil is an exception to this general classification, as it becomes oxidized before it penetrates a briquette. Experiments carried out to find an inexpensive substance that would penetrate concrete and protect it from the action of oils have so far proved unsuccessful.

G. Kroupa[4] describes a chamber for condensing flue dust from blast furnace gases, which contains a large number of corrugated curved pieces of galvanized iron, placed vertically in such a way as to force the gas current to assume a winding path, which forces it to change its direction continuously and brings it into contact with very extended surfaces before it enters the main flue. The plates being suspended vertically are self-cleaning; the dust is collected in hoppers which deliver into a pair of screw conveyors.

C. H. McDougall[5] describes, with dimensioned sketches, the plant of the Anaconda Copper Company, Anaconda, Mont., for recovering flue-dust

[1]*Engineering and Mining Journal*, 1905, LXXX, 932.
[2]*Engineering News*, 1905, LIII, p. 279.
[3]THE MINERAL INDUSTRY, 1904, XIII, 303.
[4]*Oesterreichische Zeitschrift*, 1995, LIII, 347.
[5]Canadian Society of Civil Engineers, December, 1904, through the *Canadian Mining Review*, 1905, XXIV. p. 26.

and arsenical fumes from the McDougall roasters, the reverberatory and blast furnaces, and the converters. Each division has elevated dust chambers with discharge hoppers at the bottom; from the chambers, flues varying in length (448 ft. for roasters, 1653 ft. for blast furnaces, 842 ft. for reverberatory furnaces, 703 ft. for converters), according to location deliver the gases into a V-shaped main, 60 ft. wide and 1233 ft. long, which ends in a double flue, 120 ft. wide and 995 ft. long. The stack, built on a neighboring hill, is 44 ft. square at the base, and 300 ft. high. The dust collected varies in composition. The lowest figure given is that of the blast furnace flue: As_2O_3, 1.6 per cent.; Cu, 7.4 per cent.; Ag, 6.8 oz.; and Au, 0.035 oz. per ton. The roaster dust gives higher values. The so-called converter dust is charged directly into the reverberatory smelting furnaces. The reverberatory dust averages 8 per cent. As_2O_3 and 10 per cent. Cu. At the head of the 60-ft. flue is the arsenic plant, which works up all the dust collected in the 120-ft. flue, by roasting in a Brückner cylinder. The gases from this roasting furnace pass through condensation chambers, and then enter the 60-ft. flue.

H. Rusden[1] reports results obtained in filtering the fume arising in the smelting of gold-zinc slimes by the Taverner process[2]. At the Ferreira gold mines, a flue 180 ft. long and a brick chamber had collected during a 6-months' run, in which 16,342 oz. fine gold had been reduced, 880 lb. of flue-dust containing 1.71 oz. gold (approximately 4 oz. per ton). The additional plant then erected consisted of a tubular boiler in which water circulated, a fan, a small bag-house, and a water-tower. In working up 1229.97 oz. gold, there were collected in the bag-house 70 lb. of dust, and in the water-tower 5 lb., with a total content of 0.07 oz. gold, showing that the bag-house was superfluous.

An editorial[3] gives a valuable resumé of the use of filtering devices for saving the fumes from metallurgical furnaces, taking the bag-house of the Globe works, Denver, Colo., as an example of good modern practice. The gases, arriving through the main, enter the lower brick chamber of the bag-house. Their temperature is about 120 deg. C. and may reach 150 deg. In case the temperature should rise above this maximum, cold air is drawn into the main through a by-pass. The lower chamber, divided by partitions, is separated from the upper, or filtering, chamber, by a horizontal plate from which nipples protrude upward. The filter bags are slipped over these, and tied fast. The bags usually are 30 in. in diameter, and are 30 ft. long; they are made of unbleached cotton, weighing 0.6 oz. per sq. ft., and have 46 threads per linear inch in both the warp and the woof. With gases rich in sulphur dioxide, and containing

[1] *Journal*, Chemical Metallurgical and Mining Society of South Africa, 1905, V, 288; *Engineering and Mining Journal*, 1905, LXXX, 688.
[2] The Mineral Industry, 1903, XII, p. 251.
[3] *Engineering and Mining Journal*, 1905, LXXX, 55.

an appreciable amount of sulphur trioxide, the bags are made of unwashed wool. Gases from roasting furnaces contain too much sulphur trioxide to permit filtering. A filtering area of 1000 sq. ft. per ton of ore smelted is sufficient under ordinary conditions. In the first bag-house of the Globe works, M. W. Iles gave 250 sq. ft. filtering area per ton of ore; later he increased this to 750 sq. ft. The bag-house of a lead works near St. Louis, using the ore hearth, has 900 sq. ft. filtering area per ton of ore; that of the Lone Elm works at Joplin, Mo., has 3000 sq. ft. In the manufacture of zinc oxide from ore, the bag area is 100 to 200 sq. ft. per sq. ft. of grate area; at Cañon City, Colo., where zinc-lead white is produced, the ratio is 150:1.

T. J. Greenway[1] describes the method of preparing fine material for smelting that he adopted at the works of the Broken Hill, Block 14, company, which resembles very much the one not uncommon in western smelting practice.[2] It consists (1) in mixing the fine material in a horizontal pug-mill, 8 ft. long, with water and newly slacked lime in sufficient amount to cohere to a lump when squeezed; (2) in bricking the mixture in a semi-dry brick press, the plungers of which have adjustable springs in order to be adapted to the varying compressibility of the material; (3) in stacking the bricks in kilns of 30 to 50 tons capacity, as is common with burning red brick; and (4) in firing a kiln for 10 to 20 hours, and then allowing it to burn for 3 to 6 days. The cost was $1 to $1.50 per ton of material, with labor at 25c. per hour, and wood or coal at $4 per ton.

Harmful Effect of Smelter Fume.—W. C. Ebaugh[3] gives the results of analyses from February to May, inclusive, of the air, for its content of sulphur dioxide, at two stations in Salt Lake City, and at four stations between the city and the Highland Boy, Murray, United States and Bingham smelters. They show that for 59.35 per cent. of the total time the air was free from sulphur dioxide; for 11.51 per cent. it contained one part sulphur dioxide in 1,000,000 parts air; for 20.04 per cent. from two to three parts; for 6.5 per cent. from four to six parts; for 1.91 per cent., from seven to ten parts, and for 0.77 per cent. over 10 parts of sulphur dioxide in 1,000,000 parts of air.

Lead Poisoning.—Guillemain[4] describes the changes that have been made at Mechernich, Rhenish Prussia, to render the blast furnace fumes less harmful. These changes consist mainly in replacing the bottom tap by the Arents siphon-tap with this modification, that the lead removed periodically from the well is run through a suspended iron ladle with covered discharging trough, into a row of stationary molds. A hood above the ladle is connected with a suction fan. In case the lead is to go straight to the

[1] *Engineering and Mining Journal*, 1905, LXXIX, p. 73.
[2] *Hofman*, "Lead," 1899, p. 399.
[3] *Western Chemist and Metallurgist*, 1905, I, 157, 191.
[4] *Metallurgie*, 1905, II, 74

refinery, an extension is added to the discharging trough, and the lead is run into a tilting cast-iron receiver placed on a truck. Flue-dust to the amount of 593 lb. collected in three months from the gases sucked off from the slag-pots and siphon-taps gave the following peculiar assay: SiO_2, 56.70 per cent.; Pb, 13.76 per cent.; Cu, 0.04 per cent.; Zn, 1.28 per cent.; S, 1.84 per cent.; Fe_2O_3, 6.50 per cent.

The Internationales Arbeitsamt, Bâle, Switzerland,[1] offers prizes for essays on lead poisoning which must reach headquarters by Dec. 31, 1905. While the time for the competition has lapsed, it is of value to record here the interest taken in this serious problem. The points to be covered, each by a separate discussion, are: Handling lead ores; lead smelting; chemical and electrical factories; painting, enameling, etc. The prize essays are to be published.

Desilverization of Base Bullion.

Cupellation.—K. L. Graham gives some interesting experiences in regard to the hearth material for a cupel and to the manner of ramming it.[2] Bone-ash tamped in layers cracked or peeled; tamping all the bone-ash required in one operation was a great improvement. Bone-ash being expensive in South Africa, a mixture of two-thirds of English cement, and one-third crushed fire-brick was tried, and answered fairly well; but did not stand the corrosive action of litharge for a sufficient length of time. At present Graham uses a mixture of 160 lb. of English cement, 95 lb. of mabor (mainly borate of magnesia), 85 lb. fire-brick crushed through a 12-mesh sieve, and about 40 lb. of water.

Refining Precious Metals by Means of Oxygen.—T. K. Rose[3] carried on extended experiments on the refining of gold bullion and cyanide precipitates by means of oxygen and air. The behavior of copper, lead, nickel, iron, zinc, tellurium, selenium, antimony, cadmium, bismuth, and the platinum group of metals in their connection with gold and silver was studied. The substances treated were retorted gold amalgam, zinc-box precipitate, auriferous lead, auriferous copper, and imperfectly refined brittle gold. The experiments (excepting one with air) were made with compressed oxygen, the gas being conducted to the bottom of the crucible through a clay pipe. The general results show that air or oxygen can be used successfully in certain refining operations, oxygen acting more quickly and energetically than air. Thus, base metal can be removed from gold and silver, by passing air or oxygen through them, the loss by volatilization being insignificant. Borax and silica alone are necessary as fluxes; the formula $\frac{2}{3}\ Na_2O,\ B_2O_3 + 6\ RO,\ \frac{4}{3}B_2O_3$, with R = Ca, Mg, Pb, Zn, Cu, Sb, $\frac{2}{3}$Fe, or $\frac{2}{3}$Ni, fulfills all the requirements except, perhaps, that it is expen-

[1]*Engineering and Mining Journal*, 1905, LXXIX, 1190, LXXX, 406.
[2]*Journal*, Chemical, Metallurgical and Mining Society of South Africa, 1905, V, 315.
[3]*Proc.* Institution of Mining and Metallurgy, 1904; *Engineering and Mining Journal*, 1905, LXXX, 105.

sive. As much as 90 per cent. of the borax may, however, be replaced by sand. If 20 per cent. of both the borax and the silica is omitted, the slags become pasty. The metals are oxidized in succession, according to their affinities for oxygen; the order is, however, not absolute, owing to mass-action. The paper is replete with valuable data regarding the behavior of the metals enumerated when exposed to oxidizing influences at high temperatures.

Electrolytic Refining of Base Bullion in Fluosilicate Solution.—In view of the fact that the Betts process is being introduced at Newcastle-upon-Tyne and is to be the method for desilverizing at the new De Lamar lead refinery, at South Chicago, the following papers by H. Senn[1], by R. L. Whitehead[2], and by A. G. Betts[3] are timely. The paper by Senn embodies the results of laboratory experiments with fluo-silicate solutions carried out in glass vessels of 450 c.c. capacity, with one cathode suspended between two anodes; it covers the electrolytic refining of lead, of cadmium and of lead-platinum alloys as obtained by the Deville-Debray process for treating platinum concentrate. In the series of experiments on refining lead, Senn first tested the effects of additions of gelatine to the electrolyte (containing 8.3 per cent. Pb and 11.3 per cent. of free HF) upon the cathodic lead. He found that in the absence of gelatine a current density of 1.07 amp. per sq. dcm. at 0.13 volts gave a deposit that was very irregular, and that with an addition of 0.1 gram of gelatine per liter the deposit became smooth, lustrous and sufficiently coherent to permit bending easily. Additional tests showed that the gelatine could not be decreased below 0.1 gram per liter, and that larger amounts were of no advantage. In studying the effects of current density, of course no additions of gelatine were made. With a current of 0.419 amp. per sq. dcm. at 0.05 volt, a fine crystalline deposit was obtained; but when the density was raised to 1.72 amp. per sq. dcm., the deposit became irregular and spangles were formed, reaching from cathode to anode, and causing short-circuiting after about five hours. The beneficial effect of gelatine in a fluosilicate solution is thus easily proved; but gelatine is not the sole cause of obtaining a smooth, coherent deposit. Additions of gelatine to lead nitrate and acetate solutions, while preventing the formation of spangles, and improving the character of the deposit, still gave a brittle, crumbling, cathodic lead, that of the acetate solution being inferior to that of the nitrate. Tests as to the effects of temperature showed that in a range of from -8 deg. to $+50$ deg. C. no change was noticeable. Variations in the composition of the electrolyte showed that the concentration had some influence upon the character of the deposited lead.

[1] *Zeitschrift für Electrochemie*, 1905, XI, 229, with many cross references; *Mining Magazine*, 1905, XII, 71; *Electrochemical and Metallurgical Industry*, 1905, III, 272.
[2] *Mines and Minerals*, 1905, XXV, 285.
[3] *Electrochemical and Metallurgical Industry*, 1905, III, 272.

The research on the deposition of cadmium was carried on with an anode plate of commercial cadmium and two iron cathodes, in 300 c.c. of electrolyte containing 20 per cent. of hydrofluosilicic acid, and 10 grams of cadmium sulphate ($3\ Cd\ SO_4 + 8\ H_2O$). Satisfactory results were obtained with a current density of 1.07 amp. per sq. dcm., and an addition of 0.3 gram gelatine per liter, the cadmium content of the electrolyte being enriched from 1.23 to 2.59 per cent. Summing up the results, Senn concludes: (1) That lead and cadmium can be refined electrolytically with fluosilicate solutions; (2) that an addition of gelatine to the electrolyte prevents the formation of solitary crystals and trees at the cathode; (3) that dilution of the electrolyte, or increase of the current density, interferes with the properties of the cathode metal; an addition of gelatine improves matters; (4) that the conditions for good work are: electrolyte with 11 per cent. free H_2SiF_6, and 4 to 8 per cent. Pb. (or 2.5 per cent. Cd).

The results of the experiments in refining lead, carrying varying amounts of copper, bismuth and antimony, are brought together in the subjoined table.

Anode.	Current Density Amp. Per Sq. Dcm.	Amp.	Hours Run.	Cathode Lead. Grams.	Slime. Grams.	In 25 Gr Cathode Lead.	Composition of Anode Slimes.		
							%	Cu. Bi. Sb. %	Li. O². %
Pb + 0.92 % Cu....	0.59	0.5	29	55.9	1.64	0 Cu	....	23.41 Cu	....
Pb + 0.92 % Cu....	1.07	0.9	24	83.3	0.87	0 Cu	....	36.31 Cu	24.87
Pb + 1.006 % Cu....	1.55	1.3	18	40.2	3.11	0 Cu	....	19.47 Cu	....
Pb + 1.006 % Cu....	2.30	2.0	9	69.4	1.10	0 Cu	10.3	57.96 Cu	25.12
Pb + 12 % Bi.......	0.59	0.5	24	48.3	....	0 Bi	....	70.49 Bi	....
Pb + 12 % Bi.......	1.07	0.9	7.25	25.0	12.72	0 Bi	....	42.76 Bi	....
Pb + 12 % Bi.......	1.55	1.3	11	55.1	12.60	0 Bi	....	35.44 Bi	....
Pb + 26.27 % Bi.....	1.07	0.9	17	59.0	13.25	0 Bi	12.4	23.97 Bi	1.82
Pb + 26.27 % Bi....	1.55	1.3	16.5	82.6	36.38	0.94 % Bi	12.4	60.15 Bi	1.26
Pb + 10.03 % Sb.....	0.59	0.5	30	57.9	8.58	0 Sb.	(9 % F)	67.71 Sb.	....
Pb + 10 03 % Sb.....	1.07	0.9	8.5	29.5	5.19	0 Sb.	....	47.52 Sb.	1.04
Pb + 10.03 % Sb.....	1.55	1.3	22	110.3	18.52	0.13 % Sb.	....	53.47 Sb.	....
Pb + 9.81 % Sb.....	1.55	1.3	18	90.3	10.43	0.05 % Sb.	....	45.00 Sb.	....

The electrolyte had been made up by dissolving lead fluosilicate ($PbSi F_6 + 4H_2O$) in water, adding hydrofluoric acid (HF), and 0.1 gram of gelatine per liter. It contained 9 per cent. Pb and 11 per cent. free acid. After the test, a qualitative examination of the electrolyte from the lead-copper and the lead-antimony experiments showed the presence of Pb, Sb, Fe, and H_2SiF_6; a quantitative determination gave Pb 4.83 per cent.; free H_2SiF_6, 8.56 per cent. From the lead-bismuth series a qualitative test gave Pb, Fe, H_2SiF_6; and a quantitative determination Pb, 2.48 per cent.; free H_2SiF_6, 11.5 per cent. The figures show, in both cases, that there was a strong diminution of the lead content; that in the lead-copper and the lead-antimony series the percentage of free hydrofluosilicic acid was slightly diminished, but remained practically unchanged in the lead-

bismuth series. The varying amounts of silica found in the anode slime Senn found to be due to the decomposition of the electrolyte at the anode by hydrolysis: $Pb\ SiF_6 + H_2O = Pb\ F_4 + 4HF + SiO_2$; this also explains the decrease of free hydrofluosilicic acid in the electrolyte.

The experiments with lead-platinum alloys were only partly successful. While under suitable conditions a good cathode lead was obtained, the anode slime retained lead in proportions which make it probable that it is combined with platinum, forming a compound, $PtPb_2$.

The conclusions drawn from the second series of tests are: (1) That in fluosilicate solutions lead holding all the copper it can dissolve is readily refined with a current density of 0.1 to 1.5 amp. per sq. dcm., giving pure cathode lead and a mud retaining 10 per cent. Pb; (2) that lead with 12 per cent. Bi gives pure cathode lead and an anode mud retaining 20 per cent. Pb, with a current density of 0.5 to 1.5 amp. per sq. dcm.; if the bismuth content rises to 26 per cent., the current density must be limited to 1 amp. per sq. dcm., as otherwise bimsuth is deposited on the cathode; (3) that lead with 10 per cent. Sb is satisfactorily treated with a current

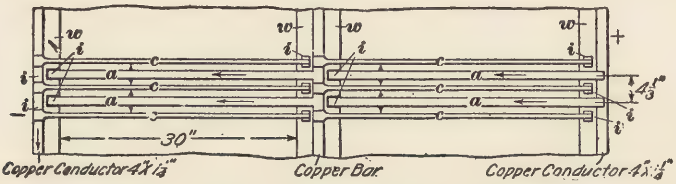

FIG. 6.—ELECTROLYZING VAT FOR BETTS PROCESS.
a, anodes. *c*, cathodes. *w*, tank wall. *i*, wood insulators.

density of 0.5 to 1.0 amp. per sq. dcm., giving an anode mud with 30 per cent. Pb; a current density of 1.5 amp. per sq. dcm. causes antimony to be deposited with the lead; (4) that lead with 10 per cent. Pt gives pure cathode lead and an anode mud consisting of $PtPb_2$; (5) that during electrolysis, lead-silicon fluoride is slowly decomposed, giving silica and lead fluoride, which go into the anode mud; (6) that the efficiency as measured by the lead deposited on the cathode is 98 per cent., but the quantities of lead going into solution at the anode always exceed 100 per cent., causing a diminution in the free hydrofluoric acid of the electrolyte.

The paper by Whitehead treats in detail the workings of the Betts process at the plant of the Canadian Smelting Works at Trail, B. C., after it had been remodeled by the author. The original experimental plant had 28 tanks, but produced only 5 tons of lead per day, at a cost of $29 per ton, instead of 10 tons, as expected. In the new plant, the old tanks were remodeled and 44 new ones were added, making a total of 72 tanks with an estimated daily capacity of 20 tons of refined lead. The tanks are 7 ft. 2 in. long, 2 ft. 6 in. wide, and 3 ft. 6 in. deep, inside measurement; they

are of 2-in. planks of selected fir, bolted together, nailed with copper nails, braced at the ends and sides with 3 by 0.5-in. iron, and painted inside and outside with P. & B. acid-proof paint. Their arrangement in two rows is shown in plan in Fig. 6. The anodes are cast from a 30-ton kettle by means of a Roesing lead-pump; they are 1 in. thick, weigh 300 lb., and are placed $4\frac{1}{2}$ in. apart. The starting cathodes, $\frac{1}{16}$ in. thick, are deposited in 30 hours on sheet steel (lead-coated and paraffined) with a current density of 4 amp. A vat holds 20 anodes and 21 cathodes, suspended from copper cross-bars, 0.5 by 1 in. The electrolyte contains 10 per cent. free HF and 5 per cent. Pb; it receives on alternate days 6 lb. glue for every 2000 lb. electrolyte, and is circulated through 14 tanks. Differences in specific gravity and composition between the top, middle and bottom of electrolyte are shown in the table given herewith:

Sample from	°B	Sp. Gr.	% Pb	% Total acid.
Top	20.5	1.167	4.03	12.52
Middle	21.0	1.171	4.23	12.60
Bottom	23.0	1.190	5.24	13.13

The fall in potential between electrodes is 0.25 volts; the current density is 15 amp. per sq. ft. of cathode area.

In the additional 44 tanks (Fig. 7), the central copper bar, 4x0.25 in.,

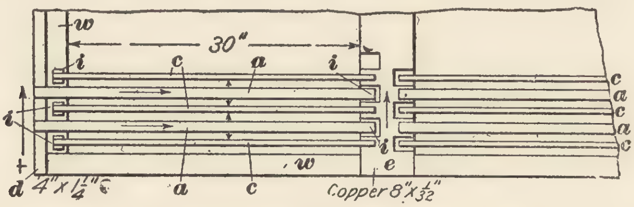

FIG. 7.—ELECTROLYZING VAT FOR BETTS PROCESS.
a, anode. c, cathode. w, tank walls. d, copper conductor.
e, copper between tanks, 8 x $\frac{1}{32}$ in. i, insulation.

has been replaced by sheet copper, $8\text{x}\frac{1}{32}$ in. thick; the electrolyte is circulated through two tanks instead of through 14, which works better. Samples of electrolyte gave the following results:

Sample from	°B	Sp. Gr.	% Pb	% Total acid.
Top	23	1.19	4.80	13.69
Bottom	24	1.20	4.95	13.71

When the lead content from top and bottom of a tank differs by more than 1 per cent., treeing takes place. A complete analysis of a sample of electrolyte gave: Pb, 5.20 per cent.; free HF, 9.50 per cent.; total HF, 13.80 per cent.; Fe, 1.34 per cent.; Al_2O_3, 0.86 per cent.; Sn, 0.008 per cent.; CaO, MgO, trace; Cu, Bi, Zn, Cd, none; K_2O, 0.056 per cent.; Na_2O, 0.261

per cent.; Sb, 0.006 per cent.; As, trace; organic matter, n.d. Daily, fresh acid and lead carbonate have to be added (see Senn above), to maintain the desired composition.

Two Holtzer-Cabot dynamos, rated at 2000 amp. and 16 volts, run in multiple, furnish through the electrodes a current of 4000 amp., at 12.5 volts. The anodes are corroded in six days, and make 22 per cent. scrap; by suspending with hooks instead of from lugs, the scrap can be reduced to 10 per cent. The cathodes weigh 150 to 200 ℔. When taken from the tanks, they are washed and brushed to remove adhering slime before they are melted down in a 30-ton kettle, poled, and cast into bars with a Roesing lead-pump, which does it in three hours. Analyses of refined lead are given in the accompanying table:

ANALYSES OF REFINED LEAD.

	Before Poling.	After Poling.	Shipments of over 500 tons.						
	%	%	%	%	%	%	%	%	%
Sb.......	0.0030	0.0015	0.0006	0.0003	0.0006	0.0003	trace	0.0003	0.0009
Sn.......	0.0410	0.0280	trace	trace	trace	trace	trace	trace	trace
Fe	trace	trace	0.0010	0.0015	0.0015	0.0017	0.0013	0.0015	0.0015
Ag,oz.p.t.	0.17	0.17	0.39	0.29	0.23	0.55	0.20	0.23	0.23
Cu.......	0.0017	0.0025	0.0005	0.0011	0.0014	0.0005	0.0010	0.0020	0.0007
Pb......	99.9533	99.9675	99.996	99.996	99.995	99.995	99.997	99.995	99.996

ANALYSES OF ANODE MUD.

Sample.	I	II	Sample.	I	II
As.........	4.91	2.2	Cu.........	1.40	6.6
Sb.........	35.71	24.6	Ag, oz. p.t.	9222.9	9397.00
Pb.........	9.57	12.6	Au.........	180.33	81.99

The treatment of slime (500 lb. in a plant of a daily capacity of two tons slime, equal to 12,000 oz. silver) is somewhat complicated, and not all that can be desired. The slime adhering to the anodes is washed off with brushes, allowed to settle, and the clear liquor returned to the electrolyzing tanks. The settled slime from the anodes and that from the tanks is freed from hydrofluosilicic acid and lead fluosilicate by washing three times with hot water in consecutive tanks, 8x4 ft., 8x4 ft. and 8x10 ft. (width not given), and placed in steps. The washed slime still retains 1 per cent. H_2SiF_6. The wash-waters are evaporated to 30 deg. B (=16 per cent. Pb and 8 per cent. H_2SiF_6), in a copper tank 8x8 ft. and 18 in. deep, drawn into a settling tank, allowed to cool, and run during the next 24 hours in small amounts into the electrolyte. The slimes, nearly freed from acid, are boiled with caustic soda (200 ℔. NaOH to 1000 ℔. slime), the solution containing 8 per cent. free NaOH.

All the antimony present as Sb_2O_3 (i.e., 50 per cent. of the total) is dissolved; it is recovered by electrolyzing the solution, twice diluted with

wash-waters, with a current density of 12 amp. per sq. ft. cathode area using iron or lead cathodes. With a lead cathode, a hard lead with 25 per cent. Sb is obtained.

The tanks (steel) are 8x3 ft. and 6 ft. deep, have oval bottoms, and are heated by two perforated steam pipes running along the bottom. The boiled slime is washed twice, sluiced into a tank of the same dimensions, but made of wood and lead-lined (12 lb.), and settled; the supernatant liquor is used for preparing the soda solution. The washed slime is now boiled for eight hours with 15 per cent. H_2SO_4, to remove copper, iron and bismuth, the necessary air being injected through the boiling pipes. Solution and wash-water are concentrated and crystalized, giving blue vitriol and mother liquor with free acid to be used over again. The slime is dried in a direct-fired steel pan, 4x8 ft. and 1 ft. deep; mixed with soda-ash and niter (400 lb. and 50 lb. respectively per ton of slime); smelted down slowly in a water-jacket cupelling furnace, lined with magnesia brick. The slag of sodium antimonate (Ag 30 oz. per ton and a trace Au) is skimmed, and the temperature is raised to oxidize any lead present, which scorifies any residual antimony and some of the copper, giving a slag which assays Ag 150 to 200 oz. and Au 0.5 oz. per ton.

After two hours' heating, the remaining bullion is free from lead, but only 0.900 fine, on account of the copper it retains. This is removed by oxidizing at a high temperature and drawing off liquid cuprous oxide (a queer proceeding when lead is available). It takes three hours to raise the fineness of the 40,000 oz. of silver-gold in the furnace from 0.900 fine to 0.950 fine. The 400 lb. of slag formed assay: Cu 40 per cent.; Ag-Au 3000 oz. per ton. They are worked into the next charge with silica, giving a slag assaying 200 oz. (an addition of silica in oxidizing the copper would save much trouble). The doré silver is cast into 500-oz. bars, and parted with sulphuric acid. The silver sulphate is decomposed with metallic copper (no precaution is taken to reduce the consumption of acid), and the precipitated silver is charged through the roof upon the hearth of a cupelling furnace, of the same general construction as given above, but holding 80,000 oz.; it is melted, covered with charcoal (to correct sprouting), and cast after half an hour in 1200-oz. bars. The gold is treated in the usual way, but melted down in graphite crucibles. Should a sample taken from the gold in the crucible prove to be brittle, the gold is refined by means of niter, working inside of a ring of bone-ash placed on the metal. The fineness is easily raised to 0.995 to 0.998.

The third paper, that by Betts, aims at solving the problem of treating electrolytic slime more satisfactorily than has been done so far. It begins by showing the behavior of the leading constituents of slime, Cu, Pb, Ag, Au, Bi, Sb, As, Se, Te, with the principal available chemical re-

agents, and then outlines the new electrolytic process, of which the following is a condensed statement:

(1) Agitate slime by means of steam and air in a lead-lined tank, with ferric sulphate solution (this sulphatizes Cu, Pb, Ag; dissolves all the Cu, and some Ag; converts As, Sb, Bi into oxides; forms $FeSO_4$ and free H_2SO_4); gives a solution (2); and residue (3).

(2) Treats the solution with metallic copper (this precipitates Ag); gives a residue to (3) and a solution free from Ag (5).

(3) Filter-press and wash the residue; then agitate with a solution containing 12 to 15 per cent. HF, and 2 to 4 per cent. H_2SO_4. This gives parting material (4) and a solution of SbF_4. (6).

(4) Melt the parting material with soda and part electrolytically with methyl-sulphuric acid (15 per cent. $CH_3.SO_3$) and 7.5 per cent. Ag. as electrolyte, and gelatine addition (1: 12,000 to 15,000); this gives Ag, Au and electrolyte charged with Cu, Bi and some Ag (1.5 per cent.). Remove Ag with Cu, Cu and Bi with Pb, separate Cu and Bi with Fe_23SO_4; reduce Bi_2O_3 to metal.

(5) Add CuO to solution, crystallize out As_2O_3 if necessary; electrolyze solution (4—5 per cent. Cu) with carbon anodes for Cu and Fe_2 $3SO_4$(to 1).

(6) Electrolyze SbF_3, regenerating HF_3, which goes with H_2SO_4 to (3).

The following analyses are given as typical of the products: Antimony; Cu, 0.07 per cent.; As, 0.41 per cent.; Sb, 99.52 per cent.; Doré silver· Ag, 78.94 per cent.; Pb, 17.56 per cent.; Au, 2.08 per cent.; Cu, 0.81 per cent.; Sb, 0.47 per cent.; As, none.

LIMESTONE.

THE production of limestone in the United States is enormous. Immense quantities are used for building material, both as cut and rough stone, and also in the aggregate of concrete. Large quantities also are crushed for road and railway material. It would be extremely difficult to collect statistics of the production and comsumption of limestone for these purposes. We are interested only in the consumption for metallurgical purposes, and to a less extent in the consumption for the manufacture of lime, which is in a way a metallurgical process.

Flux.—The chief metallurgical uses for limestone are in iron, copper and lead smelting. We are able to make a rough computation of the consumption for these purposes on the bases of the ore smelted and metal produced, especially in the case of the iron industry.

The production of pig iron in the United States in 1905 was 22,993,000 long tons. In the production of 2240 lb. of pig iron there is required an average of 1150 lb. of limestone. This indicates a consumption of 13,220,000 short tons of limestone for iron flux in 1905. The average value of the limestone used for this purpose may be reckoned at 45c. per ton.

In connection with lead and copper smelting, the bases for estimates are less well established. The bulk of the non-argentiferous lead is smelted from ore having a calcareous gangue and requiring no additional lime flux. The total production of desilverized lead of domestic origin in 1905 was 214,121 short tons. It may be assumed that an average of 10 tons of ore had to be smelted to obtain one ton of lead, which would indicate a total of 2,141,121 tons of ore smelted. We believe that this figure approximately represents the magnitude of the silver-lead smelting industry in the United States, being too low, rather than too high. Also, a certain quantity of ore of foreign origin is omitted. In American lead-smelting practice at the present time, the average requirement for lime flux is about $16\frac{2}{3}$ per cent. of the tonnage of ore smelted. This indicates a consumption of about 357,000 tons of limestone for lead smelting flux. The average value of the limestone used for this purpose may be reckoned at 90c. per ton.

With respect to copper smelting, we estimate that 652,000,000 lb. of copper derived from ore outside of Lake Superior was obtained from ore yielding an average of 100 lb. of copper per ton, indicating about 6,520,000 tons of ore smelted. Reckoning the consumption of 8 per cent. of this tonnage for lime flux, we arrive at 521,000 tons of the latter, the average value of which may also be reckoned at 90c. per ton.

The above estimates, which are admittedly rough, are summarized in the following table:

Consumed for iron flux..............	13,220,000 tons	@ $0.45=	$5,949,000
Consumed for lead flux..............	357,000 tons	@ 0.90=	321,300
Consumed for copper flux............	521,000 tons	@ 0.90=	468,900
Total..........................	14,098,000 tons		6,739,200

Lime.—The quantity of burned lime produced in the United States in 1904 was reported by the U. S. Geological Survey as 2,697,120 short tons, valued at $9,817,451. The State of largest production is Pennsylvania, which is followed in the order mentioned by Ohio, Wisconsin, Maine, Missouri, Maryland, Illinois, and New York, each of which produces upward of 100,000 tons per annum.

The cost of manufacturing lime is from $1.20 to $2.90 per 2000 ℔. of product. This corresponds to 4.2 to 10.15c. per bushel of 70 ℔. The cost per 2000 ℔. is divided as follows: Interest on cost of plant and quarry, 5 to 20c.; taxes, minor supplies, etc., 10 to 25c.; cost of quarrying two tons of limestone, 50 to 90c.; cost of fuel for burning, 30 to 75c.; cost of labor (exclusive of quarrymen), 25 to 80c. The minimum estimate represents what might be attained by a good modern plant, run steadily, and under exceptionally favorable conditions as regards quarrying, fuel and labor. The maximum estimate could easily be exceeded in a small or unsteadily operated plant.

Kilns in which the fuel is charged in layers alternating with layers of lime rock, and kilns in which the fuel is burned in separate fireplaces are both used in the United States. The former are cheaper in first cost and in operating expense, and they yield a product which is good enough for ordinary purposes. The kilns with external fireplaces on the other hand yield a white and more evenly burned product, and this type is now employed at most of the large lime-burning plants.

A modern lime kiln of the separately fired type consists of a cylindrical steel shell, say of ¼ in. plate, lined with two to three thicknesses (16 to 24 in.) of brick; the thicker the lining, the less the loss of heat by radiation. The walls of the shaft are vertical. The shaft rises from the ground to a hight of 35 to 50 ft. About 8 ft. above the ground level there is a steel platform supporting two fireplaces (on opposite sides of the shaft). The fireplaces are attended from the platform. Below the latter the shaft terminates in a steel "cooling cone," from which the burned lime is discharged by means of a draw-gate into a shallow car.

For an output of 20 tons of burned lime per day, a kiln 6 ft. in diameter and 43 ft. in hight would be required.

MAGNESITE.

The production of magnesite in the United States is derived entirely from California, and its output is all consumed on the Pacific coast; the large manufacturers of magnesite products in the East obtain their raw material from Austria-Hungary and Greece, importing it free of duty. Imports have supplied much of the larger part of American consumption of magnesite, as the following table indicates:

STATISTICS OF MAGNESITE IN THE UNITED STATES.
(Tons of 2000 lb.)

Year.	Production (*a*).		Imports.		Consumption.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1896	1,500	$11,000	(*b*)	....	....	....
1897	1,143	13,671	(*b*)	....	....	....
1898	1,263	19,075	16,039	$134,130	17,302	$153,205
1899	1,280	18,480	20,807	(*e*) 174,779	22,087	193,259
1900	2,252	19,333	28,821	(*e*) 216,158	31,073	235,491
1901	4,726	43,057	33,461	(*e*) 250,958	38,187	294,015
1902	2,830	20,655	49,786	373,928	52,616	394,583
1903	1,361	20,515	54,776	461,399	56,137	481,914
1904	2,850	9,298	38,704	286,828	41,554	296,126
1905	....	....	74,374	638,619	....	....

(*a*) Reported by the State Mining Bureau of California. (*b*) Not reported. (*e*) Estimated.

Magnesite Mining in California.

The magnesite deposits of California are found in a serpentine belt that extends along the Coast range from Oregon to southern California. It is a suggestive fact that magnesite—magnesium carbonate—is always found in association with serpentine. Serpentine is formed by the decomposition and hydration of basic magnesian igneous rocks, peridotites, dunites, etc., and in the process of alteration the magnesium carbonate appears to be evolved as a by-product, and to segregate into veins.

Napa, Santa Clara, Sonoma, and Tulare counties are the source of most of the magnesite. The operators who were active in 1905 were: American Magnesite Company, San Francisco; Sotoyome Magnesite Company, Healdsburg; and W. P. Bartlett, Porterville.

Until recently, the deposits in Tulare county have yielded the entire output of the State. They lie four miles northeast of Porterville, which town is on the Southern Pacific, 275 miles from San Francisco. W. P. Bartlett now controls the entire output. Most of the magnesite is calcined, with oil fuel, at the mine, and is shipped to the paper manufacturers at Oregon City, Ore.; the remainder is shipped in its crude state to the Western Carbonic Acid Gas Company, of San Francisco.

Magnesite occurs here in a schistose serpentine mass,[1] impregnated with

[1] "Structural and Industrial Materials of California," by California State Mining Bureau, San Francisco, 1906.

magnesite veins, containing also basalt and diabase intrusions. The present product is obtained from a hill half a mile from the kiln, where the serpentine is filled with a network of veins, in places forming nearly half of the rock mass. Some of the veins range from 2 to 8 ft. in thickness. The magnesite, being more durable than the inclosing serpentine, stands out prominently over the surface, projecting upward as much as 1 ft. in some places.

The principal development of the year occurred at the mines of the American Magnesite Company. This company holds 27 claims, 19 of which are on Red Mountain on the border between Santa Clara and Stanislaus counties, and eight on Cedar mountain in Alameda county. Those at the former locality lie 32 miles southeast of Livermore at the head of San Antone valley at an elevation of 3350 ft. Here the magnesite occurs in bold outcrops from 15 to 150 ft. wide and extending north and south for 1½ miles. Work at present is confined to the Alameda mine, which is being developed by a tunnel and a raise, opening into a quarry. The rock from the quarry will be dumped through this chute into tram-cars in the tunnel. From this an aerial tramway will deliver the magnesite to bins, from which it will be dumped into cars drawn by a traction engine to Livermore, where it will be loaded by automatic dumps into railroad cars for shipment to the company's plants at Oakland. The aerial tram has a capacity of 10 tons per hour, and the traction engine will haul three steel wagons containing 20 tons each, and will make the round trip in 24 hours. The first shipment was made in December.

Three of the company's subsidiary concerns have erected plants at Oakland on the shore of San Francisco bay, where deep-water transportation is available. The Pacific Carbonic Gas Company will make carbon dioxide for use in carbonated water and refrigeration; the Plastic Construction Company will make tiling, building material, etc.; and the Rose Brick Company will manufacture magnesite brick. These plants were all completed in 1905.

A. W. Boggs, of Riverside, is about to open a magnesite deposit which he recently purchased near Winchester. The tract is only one mile from the Santa Fé railroad.

Magnesite in Foreign Countries.

Austria.—The only producer of magnesite in Austria in the Veitscher Magnesitwerke Actiengesellschaft, of Vienna, and it is said to be the largest producer in the world. The average price per ton at the mines is $3.20 for crude and $8 for calcined. Only calcined magnesite is exported.

The amount of magnesite exported from Austria-Hungary in 1905 was 92,359 metric tons, worth 5 Kr. per ton, as compared with 53,781 tons in 1904. The United States took over two-thirds of the export in 1905.

Greece.—The two producers are the Anglo-Greek Magnesite Company, Ltd., which works the open quarries of the Monastery Galataki on Eubœa, and the Society of Public Works of Athens, which operates the underground mines Mantudi and Limni, on the same island.

Hungary.—The magnesite mines are nearly all in Gömör county and are operated by a number of public and private enterprises. The largest producer is the Hungarian Magnesite Industry Company; Alexander & Liebermann, Budapest, are the leading exporters of the material.

Transvaal.—The magnesite deposit recently found in eastern Transvaal, near the Portuguese border, to operate which, a company—Magnesite Mines of South Africa—was formed, is now being opened. An Austrian expert has lately taken charge of the works. A calcining plant, heated by producer gas, together with its complement of grinding mills, has been installed in readiness for a speedy commencement of producing operations. A cement factory is also being erected. Other products will be magnesite flooring, plate walls, tiles and carbonic acid.

Uses of Magnesite.

Crude magnesite is used for the preparation of carbon dioxide and for making magnesium sulphite and sulphate. The carbon dioxide may be obtained by calcining in kilns, after which the magnesia residue, if not needed for refractory uses, can be subjected to sulphurous gases to make sulphite; or by treating the crude magnesite with sulphuric acid, yielding sulphate. The sulphite is used by paper manufacturers for digesting and bleaching wood pulp, after which service it is neutralized with lime and used as a filler.

Calcined magnesite possesess advantages over other refractories for lining basic iron furnaces and cement kilns. The manufacturers of magnesite brick are the Harbison-Walker Refractories Company of Pittsburg, the Fayette Manufacturing Company of Layton, Pa., and the Rose Brick Company of San Francisco, above noted. Magnesia, either alone, or in combination with asbestos, forms an excellent lagging for boilers and steam pipes. The relative merits of asbestos and magnesia material have been referred to on a previous page, under "Asbestos."

MANGANESE.

True manganese ores, that is, those containing around 50 per cent. manganese, are mined in only small quantities in the United States. Iron ores, however, carrying useful proportions of manganese, are mined in important quantities. The New Jersey Zinc Company also obtains a considerable amount of manganiferous material as a residue from the franklinite ore which it treats primarily for the recovery of zinc oxide. In Leadville, Colo., large quantities of iron-silver ore, carrying manganese, but not enough silver to be workable for its precious metal, are mined and sent to the silver-lead smelters, who use it for a flux. This ore occasionally carries enough manganese to be adapted for making spiegeleisen and some of it is so employed. The following tables show the production of manganese and manganiferous ores in the United States for the last 10 years; aside from the high-grade ores, only the manganiferous ores that are used in making ferro-alloys are included:

PRODUCTION OF MANGANESE ORES IN THE UNITED STATES. (a)
(Tons of 2240 lb.)

Year.	Manganese Ores.				Manganiferous Iron Ores.				Man.Zinc Ores.	Total Production	
	California.	Georgia.	Virginia.	Other States.	Arkansas.	Colorado.	Lake Superior.	Va. & N.Car.	New Jersey.	Long Tons.	Value.
1896	318	2,538	1,588	*Nil.*	3,038	9,072	110,317	....	35,655	162,526	$339,083
1897	450	962	2,408	190	4,430	18,600	80,260	....	50,000	(b)158,600	328,176
1898	393	2,477	3,307	1,250	2,775	17,792	112,318	....	47,470	187,782	416,627
1899	263	1,623	3,626	105	855	29,161	53,702	....	53,921	143,256	306,476
1900	131	3,447	7,881	312	*Nil.*	43,393	75,360	*Nil.*	87,110	217,546	1,172,447
1901	610	4,074	4,275	3,036	*Nil.*	62,385	512,084	20	52,311	638,795	1,644,117
1902	846	3,500	3,041	90	*Nil.*	13,275	884,939	3,000	65,246	973,937	2,145,783
1903	16	500	1,801	508	*Nil.*	14,856	566,835	2,802	73,264	660,582	1,670,349
1904	60	*Nil.*	3,054	32	600	17,074	365,572	*Nil.*	68,189	454,581	789,132

(a) Statistics of 1900 and subsequent years are by the Geological Survey. (b) Includes 1300 tons of manganiferous iron ore from Vermont.

The high-grade ores from Virginia are used mainly by the chemical and glass manufacturers and brick-makers and those from California and elsewhere in chlorination works; the low-grade iron-manganese ores are used for pig iron and spiegeleisen; and the franklinite residue is smelted into spiegeleisen at the New Jersey Zinc Company's Palmerton, Penn., works, where also zinc oxide is made.

While, by comparison with the following table, domestic production appears to exceed imports by several thousand tons a year, yet the imported ore is almost uniformly of 50 per cent. grade, while that mined in this country is of much lower grade; the Lake Superior region, for example, the largest contributor to the United States production, yields ores

the richest of which carry only 20 per cent. manganese, while others fall to below 1 per cent. If the entire domestic output be reduced to terms of 50 per cent. ore, it is found to comprise less than half the amount of such ore consumed in this country; it is only within recent years, indeed, that it has supplied more than a quarter of the consumption.

CONSUMPTION OF MANGANESE ORE IN THE UNITED STATES. (a)
(Tons of 2240 lb.)

Year.	Imports.		Consumption.		Production of Man. Silver Ores (b).	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1896	31,489	$ 250,468	194,015	$ 589,551	138,079	$416,020
1897	119,961	1,023,824	278,561	1,352,000	149,562	424,151
1898	114,885	831,967	302,667	1,248,594	99,651	295,412
1899	188,349	1,584,528	331,605	1,891,004	79,855	266,343
1900	256,252	2,042,361	473,798	3,214,808	188,509	897,068
1901	165,722	1,486,573	804,568	3,130,690	228,187	865,959
1902	235,576	1,931,282	1,209,513	4,077,065	174,132	908,098
1903	146,056	1,278,108	806,638	2,948,457	179,205	649,727
1904	108,519	901,592	563,040	1,690,724	105,278	348,132
1905	257,033	1,952,407			127,170	

(a) Statistics of the Geological Survey. (b) Mined in Colorado and used as flux in silver-lead smelting; not included in the statistics of consumption.

Imports.—The United States imports manganese in ore, and in the iron alloys, ferromanganese and spiegeleisen. Manganese ore and oxide, containing not less than 50 per cent. manganese and not over 10 per cent. iron, enters free of duty; ferromanganese and spiegeleisen each pays a duty of $4 per ton. In recent years Brazil has supplied over half the entire imports of manganese ore, the rest being about equally divided among Cuba, the British East Indies, and Russia.

Baltimore, Philadelphia and New York receive over 81 per cent. of the whole imports. Most of the rest enters at Mobile for use in the steel industry of Birmingham.

Production and Imports of Iron-Manganese Alloys.—Statistics collected by the American Iron and Steel Association as to production and imports of ferromanganese (80 per cent. Mn) and spiegeleisen (20 per cent. Mn) may be represented thus:

UNITED STATES PRODUCTION AND IMPORTS OF IRON-MANGANESE ALLOYS.
(In tons of 2240 lb.)

	1903.		1904.		1905.	
	Production.	Imports.	Production.	Imports.	Production.	Imports.
Ferro-manganese	35,961	41,518	58,022	21,813	66,179	52,841
Spiegeleisen	156,700	122,016	162,370	4,623	227,797	55,457
Totals	192,661	163,534	220,392	26,436	293,976	108,298

Market.—The market for manganese ores is now practically controlled by the iron and steel industries, other consumption having fallen into comparative unimportance, while employing also only the purest and richest ores. Large supplies of manganese ore are available in readily accessible

places, those in south Russia alone, it is estimated, being capable of meeting for another century the world's demands, at the present rate of consumption—1,000,000 tons per year.

Prices for manganese ores are based on manganese contents, together with the amounts of impurities present. The United States basis is fixed by the Carnegie Steel Company. Ore must not contain more than 0.1 per cent. phosphorus, nor over 8 per cent. silica; for each 1 per cent. silica, 15c. is deducted, and for each 0.02 per cent. phosphorus, 1c. is deducted. The price per unit of manganese is 28c. per ton for ore with over 49 per cent. manganese, 27c. for 46 to 49 per cent., 26c. for 43 to 46 per cent. and 25c. per unit for ore with 40 to 43 per cent. manganese.

Russian ore of ordinary grades, during the five years 1898-1902, sold at an average price of $11.54 per ton. More recently, however, prices, particularly of high-grade ore, have advanced strongly. The price is based on 50 per cent. manganese, with phosphorus not to exceed 0.17 per cent., nor silica 9 per cent. Samples are dried at 100 deg. C., and humidity is deducted. Such ore sells at European ports for 16 to 28c. per unit of manganese ; from 5 to 10c. per ton is deducted for each per cent. of silica.

On Turkish ore, the base is 45 per cent. manganese, with limits of 0.03 for phosphorus and 11 per cent. for silica.

Japanese brown ore sells at Hamburg at from $12 per ton for 65 per cent. MnO_2 ore (41 per cent. Mn) to $27.60 for 87 per cent. MnO_2 ore (55 per cent. Mn) .

For German ore, the price is calculated on a basis of 50 per cent. MnO_2 at $4.80 per ton, with an increase of 24c. for each unit of dioxide above 50.

French ore, calcined, with 35 to 40 per cent. manganese, in 1904 brought 30c. per unit.

Manganese Mining in the United States.

California.—Merchant Bros., of Livermore, Alameda county, operate a small mill for grinding and concentrating manganese ores. It is run by steam and has a capacity of eight tons per day. One of their mines is nine miles southeast of Livermore and has been worked intermittently since 1888. The ore is soft and of high grade, occurring in contorted beds of chert which dip 45 deg. The ledge widens from 6 in. at the surface to 4 ft., where it is opened by two short tunnels. Another of their mines is the Ala Mountain, in Santa Clara county, where a 4-ft. slightly dipping vein has been exposed by an open cut and a short tunnel. The ore is shipped to Livermore.

The Fable mine, 26 miles southeast of Livermore, is likewise equipped with a mill, but no production has been reported for some time. Here a horizontal vein, 1 to 6 ft. thick, among contorted silicious shales, has been uncovered, and a short adit driven. Other deposits at which some

development has occurred are: At six miles northeast of Elsinore, Riverside county, near the Santa Fé Railway; in Los Osos mountains, in San Luis Obispo county; the Black Bear mine, near the west summit of Arroyo Nocho valley, Santa Clara county; and the Shaw mine, seven miles northwest of Cloverdale, Sonoma county.

Colorado.—Of the total output of manganiferous ores at Leadville in 1905, as already explained, 127,170 tons of ore, the silver content of which was insufficient to repay smelting costs, were used for flux by the silver-lead smelters; and 6000 tons, reaching as high as 30 per cent. manganese, were used for making spiegeleisen. Some years ago, owing to the competition of manganese ore brought as ballast from Cuba and shipped to the Illinois works, the production of this field was entirely cut off. The Colorado Fuel and Iron Company, at Pueblo, has recently entered the local market again. The Cloud City Mining Company is preparing to make extensive shipments to Pueblo. It is now in a position to ship 75 tons daily, as the two bodies, one in the Cloud City and the other in the Home Extension, have been so developed as to admit of this.

Virginia.—The Crimora mine, in Augusta county, is the largest shipper, most of its ore being consigned to chemical works. The National Paint and Manganese Company has mines in Campbell county, The ore lies in a bold hill and is reached by three tunnels 30 ft., 100 ft., and 150 ft. below the top of the hill. The 150-ft. level is the car level, and the ore mined in the upper levels and stopes is dropped down to the cars through chutes. The ore lies in a bed of yellow clay and between sandstone walls and in pockets varying from 1 ft. to 12 ft. thick, and extending from the surface down to below the lowest level, where the ore is almost solid. It is necessary to support the hanging wall by heavy timbers. As the ore is mined it is loaded on cars, which convey it to the washer. Here it is dumped from the cars into a large crusher, which reduces the large lumps to pieces not larger than an egg. It then passes through the washer, which removes all the clay and dirt, but not the rock, sand and gravel that are mined with the ore. From the washer it passes through screens, which remove all sand, small rock and gravel. It is then discharged to a platform, where boys pick out all remaining rock and gravel, and the clean ore is loaded on railroad cars and conveyed to the mill. On reaching the mill the ore is again crushed and passes to the grinding mills, where it is ground and then bolted and screened, the various sizes passing to separate bins, from which it is barreled for shipment to brickmakers. The finished product analyses 47 to 50 per cent. metallic manganese, equivalent to 76 to 82 per cent. oxide of manganese.

Manganese Mining in Foreign Countries.

Brazil.—The manganese industry of Brazil shows a constant advance.

ANALYSES OF THE WORLD'S MANGANESE ORES.[1]

Source of Ore.	Mn	Fe	SiO_2	P	Al_2O_3	CaO	CO_2	S	H_2O	Other.
	%	%	%	%	%	%	%	%	%	%
Bosnia:										
Cevljanovic, dressed ore	46.01	5.30	12.38	0.07	2.76			0.94		
Cevljanovic, dressed ore	50.42	3.53	11.48	0.07	0.90					
Brazil:										
Miguel Burnier	55.00		1.50	0.03						
Carlos Wigg	44.43	2.99	0.94	0.03	2.26	0.54			4.01	*a*4.01
Average ore	54.08	0.90	1.05	0.03						
English shipments, 1900	53.35		1.02	0.03					15.81	
United States shipments, 1900	53.46		1.11	0.03					13.00	
Chile:										
Santiago	53.00			0.05		1.13		0.02		
Coquimbo	52.66			0.06		2.33		0.05		
Coquimbo	49.79			0.02		5.36		0.63		
France:										
Vielle Aure	46.00		37.00							
La Cabesses, raw	40.42	1.75	6.50	0.04		6.00				
La Cabesses, roasted	50—56	2.00	8.50	0.05		7.00				
Saint-Giron	45.68		5.94	0.43						
Saint-Giron	56.48		6.48	0.47						
Ral el Maden, Algiers	5.89	45.16	12.29	0.09						
Greece:										
Milos	32.16	3.32	19.76	0.04	11.01	4.08	3.22			*b*0.52
Average Milos	34.73	3.00	22.92	0.06		2.15				
Manganiferous iron ore	17.37	31.10	7.1	0.04	1.61	5.42	4.00		5.9	*b*0.9
Hungary:										
Kolozvar	56.50	3.10	3.10	0.06	1.39	0.85				*c*1.63
India:										
Gosalpur	54.29	1.41	3.27	0.16	3.19				2.81	*c*2.94
Average ore	51.43	5.60	9.52	0.09						
Italy:										
Monte Porcile	48.75									
Monte Zenone	42.52		19.89							
St. Pietro, Sardinia	37.00									
Japan:										
No. 1	51.19		7.30	0.06						
No. 5	44.09		16.10	0.06						
Average ore	48.86		10.40	0.08						
Panama:										
Soledad mine	60.60	0.28	4.67	0.01	0.21	1.32			1.70	*d*0.67
Average of 23,000 tons	53.74		8.68							
Low-grade, silicious ore	47.00		30.00							
Concepcion, average of 6 ores	52.18	0.88	4.36	0.06		1.00				
Russia:										
Caucasus, high-grade	58.20	0.41	5.22	0.13	0.89	0.08	1.22	0.09		
Caucasus, low-grade	45.50		7.38	0.48					5.78	
Caucasus, average of 5 ores	54.00	0.02	4.62	0.34	0.55	0.27	0.32		1.88	
Poti	51.01	1.05	9.86	0.17						
Ekaterinenburg	53.70	0.86	8.10	Tr		1.37		0.08		*b*1.08
Ekaterinenburg	51.20	1.33	9.33	0.36		1.95		0.07		*b*0.85
Spain:										
Huelva, carbonate ore	28.26	6.58	4.95		2.11					
Huelva, carbonate ore	41.15	0.77	14.10		1.41					
Santo Domingo	38.87	1.37	22.50		1.80					
Manganese carbonate	38.33	2.31	10.85	0.10	0.35	2.87	29.88		1.54	*b*0.51
Roasted ore	49.60									
Huelva	43.70	2.51	17.10	0.22	0.90	2.38	15.40		2.19	*b*0.50
Huelva	49.15		13.00	0.10						
Cartagena	6.33	42.70	9.53	0.04	4.00	0.95	0.53	0.20	9.30	*b*0.59
Las Rosas	7.39	48.45	4.10	0.02	1.16	2.50			5.72	
United States:										
Ark. Batesville	56.92		2.10	0.10						
Batesville	54.33		1.34	0.29						
Keystone mine, Batesville	57.50	1.75	2.31	0.12						
Martin mine	60.50	1.61	1.00	0.17				0.03	10.45	
Cal. Corral Hollow	47.56	2.28	1.98							
Red Rock	27.94	3.72	35.32	0.61						
Ga. Cartersville	60.61	1.45		0.05						
Cartersville	2.30	52.02		0.24						
Cave Spring	53.44	1.98	7.79	0.06	1.52	0.08			1.56	*c*8.62
Dobbins mine	52.72	4.49	4.30	0.19						
Dobbins mine	48.83	5.40	5.05							
Dade mine	30.32	23.90	6.37	0.10						
Dade mine	43.45	4.26	16.45	0.10						
Dade mine	42.93	8.53	12.30	0.11						
Ind. Terr. Lehigh	39.67	6.15	1.45	0.07					4.75	
Lake Superior, high	12.02	47.93		0.05						
Lake Superior, low	4.89	56.35		0.05						
Va. Crimora mine	57.29	0.37		0.08						
Kendall & Flick's mine	48.25	2.70	10.50						4.00	

[1] From the compilation by W. Venator, *Stahl und Eisen*, Feb. 1, 1906, pp. 146-148. (*a*) BaO, MgO and alkalies (*b*) MgO. (*c*) BaO. (*d*) CuO.

Exports during 1905 were 224,377 tons as against 208,260 tons for 1904, the largest annual output previously reported.

The most important deposits in Brazil are found in the following districts: (1) Minas Geraes (Miguel Burnier mine); (2) Lafayette, or Queluz, near Ouro Preto (Barrosa, Morro da Mina, Piquery, Sao Gonçalo mines); (3) Bahia, at Nazareth, near San Salvador; (4) Matto Grosso, south of Corumba; (5) Amazon district (Macuara and the Nhamunda mines). The deposits are mostly at a great distance from the coast, but can be worked at a profit. The deposits in Minas Geraes, Lafayette, Bahia, and Matto Grosso are of special importance. Well-known mines are situated in the district between Miguel Burnier and Ouro Preto—Rodeio, Capao, Rodrigo, Silva, Saramenha, Bocaina, Vigia, Ressaquinha, and Ilhecs.

The Morro da Mina Manganese Company in the year ending June 30, 1905, quarried 65,753 tons of manganese ore and exported 78,067 tons, as against 86,032 tons extracted and 66,362 tons exported during the previous year.

The manganese deposits recently discovered at Nazareth, Bahia,[1] are similar in their geological formation to those at Queluz, in Minas Geraes, but differ from the manganese deposits in other parts of the world. The ore occurs in the form of veins, dikes or lenticular masses in crystalline rocks of various types, which may be included under the generic name of gneiss. These rocks constitute the base of the formation and are cut by dikes, concretionary masses, and nodules of eruptive rocks of different types, among which are granite (this predominating) and the type of rock known as queluzite, which contains the manganese.

Queluzite consists principally of a primitive oxide of manganese and a mixture of silicate of manganese and alumina. The orebodies have an inclination of over 45 deg. When the surface deposits are exhausted, which are the only ones now being worked, underground mining may be profitably carried on. The surface indications and the work already done give ground for the belief that the manganese mining industry will become permanently established in this district.

British North Borneo.—Manganese ore has been found close to Taritipan, near the southern end of Marudu bay, on the northwest coast of British North Borneo.[2] Outcrops of the ore, generally with but light covering, have been met with over an area of 12 sq. miles. The ore is mainly psilomelane, but of unusual hardness; even in the rainy season it absorbs only 6 per cent. moisture. Analyses of samples from various parts of the deposit indicate that average shipments would contain manganese dioxide equivalent to from 49 to 51 per cent. metallic manganese, 15 per

[1] Orville Derby, *Boletim* of the Agricultural Department of the State of Bahia, March, 1905.
[2] *Bulletin* of the Imperial Institute, 1906, II, No. 4, p. 309.

cent. silica, 0.35 per cent. sulphur, and 0.03 per cent. phosphorus. The silica is rather high, but the sulphur and phosphorus decidedly low. A picked cargo would contain 8 to 10 per cent. silica and the equivalent of 51 to 53 per cent. manganese. Of the regular output about 35 per cent. would fall into the latter class. It is estimated that an annual export of from 40,000 to 50,000 tons can be easily maintained.

Cape Colony.—Manganese ore is reported in the Constantia valley, the Paarl district, 36 miles from Cape Town on the main railway line to the Transvaal. The ore is stated to compare favorably with that of southern Russia. An analysis of the ore taken from the surface showed 71.5 per cent. of manganese dioxide. The Cape Manganese Ore Company, Ltd. (capital £75,000) was formed, Sept., 1905, to work these deposits. Their development will stimulate the iron and steel industries of South Africa.

Colombia.—The manganese deposits of Colombia have been worked during late years to a considerable extent; the ores are high-percentage manganese with little phosphorus and silica. The most important deposits are in Panama, in the neighborhood of the port of Nombre de Dios. The more important mines are Viento Frio, Culebra, Cavano, Concepcion, La Guaca, and Solidad, which are worked by the Caribbean Manganese Company and the firm of Brandon, Arcas & Fillipi.

Cuba.—The island of Cuba has produced of late years considerable quantities of manganese ores, which have been won in the mines of Tampo and Cristo, in the Santiago district. The production varies; in 1902 40,000 tons, and in 1903 only 21,000 tons were produced. The Cuban deposits are of much importance to the North American steel industry.

Germany.—The annual production of Germany being only about 50,000 tons the several mines have each only a relatively small output. Attempts are often made to engage large companies in manganese mining, but capitalists avoid it, though the iron works would absorb the whole production, providing that the ore satisfied the requirements. Up to the present no noteworthy results have been attained, and the opinion is that the deposits are mostly irregular. The pulverulent character of the Hessian ores renders them unsuitable for the preparation of high-grade ferro-manganese alloys.

The firm Stumm Bros., of Neunkirchen, give the following analysis of the ore from their Eleonore mine, near Rodheim-on-the-Bieber: Iron 26 to 28 per cent.; manganese, 15 to 17 per cent.; residue, 22 to 23 per cent. Ores from Weiler and Waldalgesheim contain, according to Wandesleben Bros.: Iron, 28 to 30 per cent.; manganese, 18 to 20 per cent.; and little residue. Ores won in the Lahn district during 1904 had the following average composition: Iron, 28 per cent.; phosphorus, 0.06 to 0.12 per cent.; manganese, 15 to 51 per cent.; silicic acid, 5 to 15 per cent.

Ores from the Oberrossbach mines, which formerly supplied the greatest

quantities, have the following composition: Iron, 25 to 26 per cent.; manganese, 24 to 25 per cent.; residue, 11 to 59 per cent.; while the best manganese ore contained from 45 to 90 per cent. of manganese oxide.

Formerly, high-percentage manganese ores were won at the Kreis Biedenkopf mines, near Laisa. The raw ore contains from 15 to 25 per cent. manganese. High-percentage ore (over 50 per cent. manganese) is recovered by hand separation. The ore is of compact structure and high specific gravity, with 10 to 12 per cent. silica.

In 1904, the total output of the mining district of Bonn was 47,110 tons. In the mining district of Weilburg, manganese ores were recovered partly from the slimes of old ore dressings, partly from the Obertiefenbach and Steinbach mines. In the mining district of Coblentz-Wiesbaden, the Kous Schlossberg mine, near Johannisberg, in the Government district of Wiesbaden, yielded 625 tons of manganese ores. The four mines in the Government district of Coblentz—Amalienhoehe, near Waldalgesheim; Concordia, near Seibersbach; and Elisenhoehe-Waldalgesheim, near Bingerbrueck—produced together 50,407 tons.

Since 1900, the Nora Mining Company has undertaken extensive prospecting at the Laisa mine, which has proved the existence of regular manganese ore deposits, in veins of great length, and a considerable output will be possible. These deposits will gain in importance when the projected railway traverses the district. The large amount of silica found in dressing experiments is discouraging. A part of the ores should be suitable for the preparation of ferro-manganese, and another part as an addition to the oolitic iron ores of Loraine and Luxemburg.

Production of Germany in 1905 was 51,463 metric tons as against 52,886 tons in 1904. Imports of manganese ore were 262,311 tons in 1905, an increase of 6551 tons. Exports were small, 5536 tons in 1904, and 4116 tons in 1905.

Greece.—The most important deposits are situated in the island Milos, near Cape Vani, Fourkorouni, and in the island Andros. The output in 1902 was 15,000 tons. The Grecian deposits will probably be of considerable importance in the future, and the industry is progressing.

India.—The most important deposits are in the following districts: (*a*) Vizagapatam (500 miles north of Madras); (*b*) Kamptee (Nagpur); (*c*) Bhandara and Balaghat; also in Central India, in the State of Thabua, near Ratlan, and at Viziniagram. The Indian ores have to pay high railway rates. The percentage of manganese is, as a rule, lower than that of the Caucasian ores, and the ores are more difficult to reduce.

The rapid development of manganese-ore mining is at present the most conspicuous feature in the mineral industry of the Central Provinces. There are now 15 mines and sundry shallow workings in operation. The yield of the mines in 1905 is officially stated at 85,034 tons, but these

figures, it is said, do not include the output from shallow workings, which are sometimes of considerable importance.

The production of India in recent years is given in the following table:

District.	1903.		1904.	
	Long Tons.	Metric Tons.	Long Tons.	Metric Tons.
Central India.				
Jhabua State	6,800	6,909	11,564	11,749
Central Provinces.				
Bálaghát district	7,898	8,024	10,323	10,489
Bhandára district	....	....	8,558	8,695
Nagpur district	93,656	95,159	66,153	67,214
Madras.				
Vizágapatám district	63,452	64,470	53,699	54,260
Total	171,806	174,562	150,297	152,707

The Indian Geological Survey has made an examination of the manganese deposits in the Central Provinces, covering the districts of Nagpur, Chhindwara, Bhandara, Balaghat, Mandla, Seoni and Jubbulpore.

Braunite is the predominating oxide of manganese in the orebodies which are being worked. These occur as lenticular masses lying in the ordinary gneiss, schist and quartzite of presumably Archean age; they appear to have been formed, at least partly, by the alteration of manganese silicates, among which spessartite, rhodonite and manganese-pyroxenes are prominent. Manganiferous silicates—piedmontite, spessartite and mangan-hedenbergite—are found also in the crystalline limestone of the Nagpur and Chhindwara districts, where they have been changed in some instances into oxides, which would be considered worth working but for the large quantities of richer ore exposed in the same area.

In the four districts grouped together near the eastern edge of the Deccan Trap (Nagpur, Chhindwara, Bhandara and Balaghat) over 60 manganese deposits have been described, most of which contain material pure enough to command a price sufficient to cover the cost of mining and transport to European and American markets. The higher grades only are being exported; the amounts which have been quarried have made no serious impression on the total stocks in sight.

Compared with the previous year, there appears a decrease of 32,490 tons in the output of this mineral for 1904. In spite of this, the industry shows no signs of languishing. Manganese is quarried, not mined; the output for quarries under 20 ft. deep is not included. The export of manganese ore from India during 1904 was 154,829 tons.

A Bombay firm has acquired the right to work a tract of the southern Mahratta country in which manganese ore is found. The advantage is

not so much in the superior quality of the mineral as in the short haul to shipboard. It will be exported from the port of Marmagoa.

Italy.—Deposits are found in Liguria, near Gambatersa, Monte Porcile, and Monte Lezone; in Tuscany, near Rapolana and Monte Argentario; on San Pietro Island, southwest of Sardinia; in Piemont, San Marcel. Although the existence of 3,000,000 tons of ore has been estimated in Tuscany and Liguria, the production is small on account of the poor quality of the ore.

Russia.—The productive areas are the Caucasus, South Russia, and, to a less extent, the Urals.

Caucasus.—In the government of Kutais, at Tschiatura, in Sharopon, the deposits are of a thickness of 5 to 8 ft., extending over a surface of 30,000 acres, and have produced, from 1848 to 1904, 4,322,600 tons of manganese ore, with an average composition of 50 per cent. Mn, 6 to 8 per cent. SiO_2, and 0.12 to 0.17 per cent. P. The ore consists of pyrolusite and manganite of oolitic structure, and is very friable. The raw ore is separated by hand, yielding 33 per cent. of ore for shipment. There are 5000 holdings in the Province of Kutais, which belong to small companies, peasants and merchants. The ores are shipped from Poti. The average freight to England is $3.20. From the deposits, the ore is transported 1 to 6 km. in mine trucks or carts to Tschiatura; from there to Sharopon, 40 km., by narrow-gage railway; and, lastly, 131 km. by railway to the harbor of Poti. The costs of ore at Poti are as follows:

Cost of production at mines	$0.95
Transport from the mine to Tschiatura	1.00
Transport from Tschiatura to Sharopon	3.54
Transport from Sharopon to Poti, unloading, stores, etc	1.00
Freight to England, sampling, and insurance	3.31
Total	$9.80

South Russia.—In the government of Ekaterinoslaw, Nikopol district, on the river Tomakovka, 16 versts south of Nikopol, the deposits near the small town of Nikolaiovka have been worked since 1886. Up to 1904 about 750,000 tons had been extracted, and the reserve of ore is estimated at 7,500,000 tons. The ore consists of pyrolusite in lumps, with little dust. The cost of transport is considerable, the ore having to be carted for a distance of about 30 km. and then by small boats to the smelting works at Alexandrowsk, or to the harbor of Nikolaieff, for export. The South Russian exports are at present small in comparison with the exports via Batoum and Poti. A railway is projected, and, when completed, South Russian will be able to compete favorably with Caucasian ores.

Of all known manganese ore deposits the Russian are the most extensive, the thickest, and the most regular. According to estimates the reserve of manganese ore is: In the Caucasus, 98,000,000 tons, and in Nikopol

40,000,000 tons. No estimates are available of the deposits in the Ural district. Assertions concerning estimated quantities of ore are very diverse; in any case, a large and regular output in the Caucasus is to be reckoned upon. The output of spiegeleisen and ferro-manganese has only attained modest proportions in Russia. In spite of an import duty of 50 copecks per pud on spiegeleisen and 75 copecks on ferro-manganese Russia continues to draw her supplies of manganese alloys from England and Germany.

The methods of mining manganese ore in Russia are still of a primitive character; the ore is not put through any mechanical cleaning process, it being simply classified into medium, rich, and very rich. The poorest ore mined does not contain less than 20 per cent. of metal, while the average is about 50 per cent., so that concentration is not much required. There are over 400 small proprietors, working with insufficient means and without any foresight. It is, therefore, difficult to arrive at any working agreement, and all the large undertakings that have endeavored to introduce a more rational system of working have succumbed under the competition of the small owners. In fact, the only sphere in which large capital has been able to intervene in this industry is in the hands of middlemen between the small producers and the market.

Once the ore has been roughly classified, it passes immediately into the hands of middlemen and commission agents. Since the Government succeeded in constructing a narrow-gage mining railway between Sharopon and Tschiatura, a distance of 31 miles, the conveyance of the ore has become comparatively easy, since it can be loaded into cars almost directly from the mines. The cost of the 50 per cent. ore delivered to the port of Batoum is 46c. per 100 ℔., of which the transport to Batoum constitutes half.

Attempts have been made from time to time to regulate the price of Russian ore, but up to the present no modification has taken place in the general condition of the Caucasian industry. In 1905 the situation was complicated by labor troubles at the mines and ports of shipment. Faced by keen competition and a loss of markets, the producers have proposed to ask the Government to lower the railway rates, more particularly on the Tschiatura branch. To recoup itself for the high outlay on this line, the Government fixed the rate for the 31 miles at 7½ copecks per pud (about $3.54 per ton) of ore, after having at first charged 10 copecks. The mine owners' application for a further reduction has not been granted. All attempts to secure a reduction of railway rates are opposed by the mine owners in Nikopol, in the Province of Ekaterinoslaw.

Sweden.—The most important mines are situated near Undenäs (Bölet), West Gothland; Spexeryd, Hohult, Jacobsburg, Ludwigsburg in Smaland;

Skidburg and Nalburg in Leksand; Langban, and Paisburg in Wermland. The production of rich ore is small.

Spain.—The Spanish manganese mines are now nearly exhausted. The best known deposits are situated in Huelva, Ciudad Real, Ovideo, Teruel and the mines Asturiana Magenta, Mecurio, Maude, and Excelsior in North Spain. The production diminishes from year to year. In Teruel several new deposits have been discovered.

Turkey.—Macedonia and Asia Minor furnish large quantities of manganese ores. The Cassandra deposits are the most noteworthy, producing about 60,000 tons yearly. The ore contains on an average 2.45 Fe, 44.83 Mn, 0.012 P, 9.40 SiO_2, 6.18 per cent. CaO.

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*)
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Bosnia (*b*)	Brazil. (*d*)	Canada.	Chile. (*d*)	Colombia.	Cuba.	France	Germany.	Greece.	India.	Italy.
1895.....	7,733	22,478	8,145	5,490	113	24,075	6,025	....	30,871	41,327	7,250	16,070	1,569
1896.....	5,941	23,265	6,821	14,120	112	26,152	10,668	....	31,318	45,062	15,500	57,783	1,800
1897.....	10,043	28,372	5,344	16,054	14	23,528	8,382	....	37,212	46,427	11,868	74,862	1,634
1898.....	14,219	16,440	5,320	26,417	45	20,851	11,176	....	31,935	43,354	14,097	61,469	3,002
1899.....	10,484	....	5,270	65,000	279	40,931	10,160	....	39,897	61,329	17,600	88,520	4,356
1900.....	14,550	10,820	7,939	108,244	34	25,715	8,748	21,973	28,992	59,204	8,050	129,865	6,014
1901.....	12,077	8,510	6,346	100,414	447	18,480	95	25,586	22,304	56,691	14,166	122,831	2,181
1902.....	12,883	14,440	5,760	157,295	175	12,990	*Nil.*	40,048	12,536	49,812	14,962	160,311	2,477
1903.....	11,489	6,100	4,537	161,926	135	17,110	(*c*)	21,070	1,583	47,994	275,232	174,563	1,930
1904.....	*f*10,189	485	1,114	208,260	123	(*c*)	(*c*)	33,152	(*c*)	52,886	239,635	140,953	2,836
1905.....	....	....	......	....	22	....	.....	....	......	51,463	228,182	......	.....

Year.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	South Australia. (*d*)	Spain.	Sweden.	United Kingdom	United States.
1895......	17,141	213	1,240	361	203,081	49	10,162	3,117	1,293	173,237
1896......	17,967	66	1,494	305	208,025	*Nil.*	38,265	2,056	1,097	165,135
1897......	17,351	182	1,652	403	370,195	*Nil.*	100,566	2,749	609	161,138
1898......	11,517	220	907	68	329,546	*Nil.*	102,228	2,358	235	190,787
1899......	11,340	137	2,049	747	659,301	102	104,974	2,622	422	145,548
1900......	15,228	166	1,971	77	802,234	*Nil.*	112,897	2,651	1,384	221,714
1901......	16,298	208	904	221	522,395	134	60,325	2,271	1,673	649,016
1902......	10,866	*Nil.*	*Nil.*	4,674	536,518	18	46,069	2,850	1,299	989,519
1903......	5,616	71	30	1,341	*d*458,894	10	26,194	2,244	831	671,151
1904......	(*c*)	199	1,851	843	(*c*)	(*c*)	18,732	2,297	8,880	461,854
1905......	....	....	....	....	....	....	....	....	14,582	....

(*a*) From official statistics. (*b*) Includes Herzegovina. (*c*) Statistics not available. (*d*) Export returns. (*e*) Includes output of manganiferous iron ore. (*f*) Austria only.

MICA.

IMPORTS from Canada and India still supply the larger part of the mica consumed in this country, partly because the domestic production is narrowly limited as to locality and quantity, but more especially because domestic mica does not meet the requirements of the electrical industry, which is now the chief consumer of the mineral. Imports pay a duty of 20 per cent. *ad valorem,* in addition to 6c. per lb. on unmanufactured and 12c. per lb. on cut and trimmed mica. The failure of domestic producers to take advantage more fully of this vigorous protection can be explained only by the decreasing importance of the industries that require mica in large sheets, for which, alone, high prices are realized. The growing consumption of scrap mica is fully satisfied by the refuse from the sheet-mica mines, so that it is unprofitable to open a deposit unless it will yield 10 per cent. of sheet-mica; no reliance whatever should be placed upon an output of mica suitable only for grinding.

STATISTICS OF MICA IN THE UNITED STATES.
(In pounds and tons of 2000 lb.)

Year.	Production. (a)			Imports.			
	Sheet. (c)	Scrap.		Manufactured.		Cut or Trimmed.	
	Pounds.	Sh. Tons.	Value.	Pounds.	Value.	Pounds.	Value.
1896	17,630	570	$7,687	....	$169,085	....	....
1897	118,852	2,882	28,820	(b) 722,939	161,334	226,771	$41,068
1898	110,918	3,529	39,837	877,930	115,930	78,567	34,152
1899	97,586	6,917	50,596	1,709,839	233,446	67,293	42,538
1900	127,241	5,417	42,889	1,892,000	290,872	64,391	28,688
1901	360,600	2,171	19,719	1,598,722	299,065	78,843	35,989
1902	373,266	1,400	35,006	2,149,557	419,362	102,299	46,970
1903	619,600	1,659	25,040	1,355,375	288,783	67,680	29,186
1904	668,358	1,096	10,854	1,085,343	241,051	61,986	22,663
1905	....	....	....	1,506,382	352,475	88,188	51,281

(a) Statistics for 1901 to 1904 inclusive are those of the Geological Survey. (b) The classification required by the Dingley Act went into force July 24, 1897. The figure for unmanufactured includes the entire unclassified import of that year previous to July 24; the figure for cut or tri med covers the import of that class during only the last five months of the year. (c) The value of sheet mica being so widely variable, and so little indicative of commercial results, and all previous statistics being of doubtful accuracy, they have been omitted from this table.

Uses.—The manufacturers of electrical machinery are the largest consumers of mica. It is as an insulator between the segments of commutators that the mineral is most largely employed. The most valuable quality in mica designed for this purpose is softness, whereby it wears down flush with the copper segments, under the friction of the brushes. The General Electric Company supplies its needs from its own mine near Sydenham, Ontario. In big armatures, the expensive large single sheets formerly required are now replaced by built-up pieces, "micanite," in which small and inexpensive sheets are cemented together, by a patented process, to make sheets of any needed size. The necessity for the large

sheets formerly used in stove doors has now been obviated by framing the doors in small panels. The use of mica lamp chimneys and other novelties, however, still maintains some demand for large sheets. Scrap mica, refuse from the cutting and trimming works, is now ground to various degrees of fineness and used for boiler and pipe lagging, for roofing and fire-proofing materials, as a lubricant, and for decorative work, wall papers and paints.

Mica Mining in the United States.

All the mica mined in the United States is muscovite, the white, potash variety, except some lepidolite, which is used only as a source of lithia salts. The only States that maintain regular output are Idaho, New Hampshire, South Dakota and North Carolina.

Idaho.[1]—The Robinson mining district, in Latah county, 18 miles from the Northern Pacific Railway, was exploited in 1895, since which time increasing quantities of mica have been taken out. It is found in a pegmatite vein averaging 6 ft. wide which has been traced for over a mile. Crystals containing plates 22 by 26 in. have been taken out. It is notable that this is one of the few places in the world where mica has been mined in a systematic modern fashion. An adit 600 ft. long has been driven to the lode, and a 500-ft. raise.

New Hampshire.[1]—This State, up to 1869, was the only commercial source of mica in the United States. The mica district extends through the western portion of the State from about the central to the southwest extremity, in Grafton, Sutherland and Cheshire counties. Mica has also been mined at Danbury, in Merrimac county, just south of the Grafton county line. The most important mines for many years were the Ruggles at Valencia and Palermo, the first of which was opened in 1803, and furnished for many years the bulk of the country's supply. About $8,000,000 worth of mica has been taken from this mine. For the last 15 years the mines have been worked only fitfully. Only the sustained demand for scrap-mica has kept the industry alive. Most of the product has been shipped from Bristol, Canaan, Grafton, Rumney and Warren, in Grafton county, from Danbury in Merrimac and Alstead in Cheshire county.

South Dakota.—Mica of commercial value was found in the Black Hills in the early nineties, and the deposits have been worked to a small but continually increasing extent up to the present time, notwithstanding the fact that the sizes are small and the quality of less value than the Eastern mica. The principal district lies on French creek 6 miles east of Custer City. The New York mine, near Custer, is the principal one in this district. Further north, near Deadwood, there are other mines, of which the Lost Bonanza is the most important producer. The Crown and Daly mines were opened in 1902.

[1] G. W. Colles, *Journal* of the Franklin Institute Nov 1905.

North Carolina.—This State now affords over three-quarters of the country's production of mica. The productive areas are in the mountain ranges along the western boundary and also to the east of the Blue Ridge; the former are the more productive. With few exceptions, the mica deposits are worked by individuals and generally on a small scale, for a few days to a few months during the year, obtaining mica valued at from $10 to $500 as a yearly production. There are, however, a few large companies which are carrying on systematic mining, sinking their shafts below water-level and equipping their mines with steam hoisting and pumping plants. Some of these shafts are down between 400 and 500 ft. The principal companies operating are the Burleson Mica Company, in Mitchell and Yancey counties; the Sylva Mining Company, in Haywood, Jackson and Macon counties; and the Laytown Mica Company, in Caldwell county.

Mica Mining in Foreign Countries.

Canada and India are the only regular producers of mica in large amounts, and their product is almost exclusively phlogopite, the brown or magnesian variety.

Canada.—Ontario and Quebec are the staple producers. In both Provinces, muscovite, the white mica, is known to occur, and it has been mined in Quebec, but in neither has it ever acquired any commercial importance. It has also been found in northern British Columbia, in a region remote from transportation.

Ontario.—The mica deposits are every year coming more under the control of the large electrical machinery builders, who mine their own supplies, but put little mica on the market. Since the success of micanite and other means of utilizing small mica has become assured, many old dumps have been worked over and regular mining has been retarded accordingly; the dumps are now nearly exhausted, and productive mining will progress once more.

The General Electric Company, Schenectady, N. Y., operates the Lacey mine in Loughboro township and the Hanlan mine, near Perth, in Burgess township. The shaft of the former is 200 ft. deep and six levels are driven on veins. The surface is equipped with a boiler and compressor plant. The company is exploring with the diamond drill on contiguous territory. In Ottawa, the company operates a dressing plant, employing 90 workers to trim and cleave the crude mica; the entire output of the company's mine is prepared here for shipment to the United States.

The Laurentide Mica Company operates a mica factory in Ottawa, obtaining its supplies of crude from its own mines at Chelsea, Quebec, and from independent producers in both Provinces. Its product is shipped to the Westinghouse company, Pittsburgh, Penn. Other mica dressing works are those of Eugene Munsell & Co., New York, at Ottawa,

Kent Bros., at Kingston, and Montgomery & Adams, at Perth, all of whom have mines of their own, but get additional supplies from small independent producers. Other mica miners, not already mentioned, are Richardson Bros. and Mills & Cunningham, both of Kingston.

Quebec.—The mica mines of this Province have suffered more severely by the recent stagnation. The principal operators are: Blackburn Bros., Perkins' Mills; Wallingford Mica and Mining Company, Perkins' Mills; Fortin & Gravelle, Gratineau; Kent Bros., and the Laurentide Mica Company, already referred to. The output from Quebec in 1905 comprised 357,160 lb. of thumb-trimmed, worth $89,060, and 21,400 lb. of scrap, worth $6,400.

India.—Exports of mica from India, during the fiscal year ending June 30, 1905, amounted to 1,957,500 lb., valued at £97,932, or $0.24 per lb., as against 2,154,800 lb., worth £86,296 in the preceding year, showing a marked improvement in the selling price. Bengal exported 1,316,700 lb., Madras, 633,400 lb., and Bombay, 7400 lb.; the Madras output was the most valuable, worth $0.29 per lb. The value of the Indian product in that year was greater than the combined value of the productions of Canada and the United States.

There are two principal regions[1]—Behar in Bengal, and Nellore, in Madras, on the east coast. The small export from Bombay comes probably from Coorg, a small province in southwest India.

The Behar region has three principal districts, Hazaribagh, Gaya and Monghyr. In the first, some 250 mines have been opened, of which about 60 or 70 are now operated. The principal mines are Dumcho-Gharanchi, Bochagta, Selboya and Kodama. They are from 35 to 70 miles from the nearest railway. The mica dikes are largely decayed and disintegrated for a distance of 100 ft. and upward, which is no doubt attributable to the excessive rains. There is but one white mica mine in Bengal (the Singur mine), and in spite of its more frequent occurrence the phlogopite mica is valued at twice the price of white mica. In Gaya the principal mines are Singur (white mica), Vita Chatkari, Bind, Govinpore and Korarama. The largest plates reported from the Bengal mines measure about 18x24 in.

The Madras mica industry is practically new, the mines of Nellore having first become producers for export about 1895. The Nellore district in the coast region north of Madras was the first to be opened, and increased rapidly in importance. Since then two other mica districts have been opened, North Arcot to the west of Madras, and Nilgris in the southwestern part of the peninsula. Some of the Madras mica is of immense size, plates 5 ft. long and 40 in. wide having been taken from the Itakuri mine of the Nellore district.

In Bengal the mines are worked only during the six dry months of the

[1] G. W. Colles, *op. cit.*

year, from November to May, when there is no agricultural work. Mining is carried on in the crudest way by the farmers of the surrounding country and economy in operating is not attempted. Open cuts are made along the outcrops of the dikes where mica is seen in quantity, and are continued down 20 to 30 feet until the sides become dangerous. No timbering is used and fatal accidents are of frequent occurrence. The mining tools and methods are so crude that it is not profitable to carry the pits further than the dike material has been sufficiently disintegrated to make progress easy. The mining is done by coolies, both men and women, whose wages average 4c. a day for the men and 2c. for the women.

The shafts are usually on an incline and sometimes have a diameter of 15 to 20 ft., but most are just big enough to allow several pairs of miners to get in and wield their hammers in a cramped position. The tools used are a drill, a chisel and a hammer; no explosives are used.

Some of the larger mines run to a depth of 100 and even 150 ft. In these cases small vertical ventilating shafts about 2 ft. in diameter are put down to ventilate the inclines, and serve as means for raising the excavated material, for which purpose they are provided with small wooden lifts.

In the Hazaribagh district the mines on the Dórunda and Satgawan estates, and those in the Koderma Government forest reserve are under Government management, and are leased by the Government at R. 50 per acre annually, with certain provisions which practically prevent speculation and compel their operation by the lessee. Of the private estate mines, the majority are operated by F. W. Christein & Co., Tisri, who in 1897 operated 110 mines; the next largest operators were Raj Krishna Sahana (31 mines) and W. R. McDonald (28 mines), both of Koderma.

Work is carried on only in the daytime, from 8 A. M. until dusk. At the end of each day's work the product is gathered in bundles, and taken to the manager's bungalow, ready for the trimmers and cutters the next day.

The splitting, trimming and cleaning is done as follows: The workman holds the blocks on a peg projecting 18 in. above the ground and opens them with a knife into slabs ⅛ in. thick. The ragged edges are then trimmed and the plates assorted according to quality and size. They are then packed in 100-lb. boxes and carried to the railroad 30 to 100 miles distant. All the Bengal product is exported from Calcutta.

In Madras a more modern system is employed. The mica mines of Nellore are on Government land and are leased to individuals.

MOLYBDENUM.

Molybdenum is one of the several minor metals that have come into prominence through the valuable qualities that they impart to special steels. Except vanadium it is now the least commonly used, partly because its supply is limited, but more particularly on account of certain metallurgical difficulties to which reference will be made. Molybdenite, the sulphide, MoS_2, and wulfenite, the molybdate of lead, $PbMoO_4$, are the only molybdenum ores found in commercial quantities in this country. Neither of them is produced with regularity, although discoveries of these ores are constantly being made and developed, and already two mills have been equipped for treating them.

The distinctive characteristics of molybdenite were described in our last volume. In appearance, molybdenite closely resembles flake graphite, the only other mineral with which it is liable to be confused. It is, however, twice as heavy as graphite and its streak on white paper has a distinct greenish tinge, while that of graphite is clear black. Any one of several simple blow-pipe tests will distinguish sharply between the two minerals. Molybdenite has been found in commercial quantities only in the more acid igneous rocks, generally granite, and usually in quartz or pegmatite veins.

Molybdenum Mining in the United States.

Arizona.—Wulfenite occurs in the copper mines along the Rio San Pedro, Pinal county. The dumps of the old Mammoth mine have yielded a substantial output of wulfenite, and the Troy-Manhattan Copper Company has built a 40-ton concentrating plant to treat the wulfenite ore from its copper mine at Troy.

California.—Three molybdenum deposits have recently been located by E. E. and T. S. Grider, of Porterville, Tulare county, about two miles south of Nelsons, in the same county. T. W. Webb has located eight claims of molybdenum ore in the west arm of Death Valley, south of Lida, Inyo county.

Colorado.—Molybdenum ore is found in Boulder county; also in Lake county.

Idaho.—A discovery of molybdenite is reported on the south fork of the Salmon river, 14 miles from Warren.

Maine.—The American Molybdenum Company, of Boston, has opened a

molybdenite deposit at Cooper, Washington county, and has built a concentrating mill. This locality is advantageously situated and ought to become a regular producer.

Concentration of Molybdenite.—Molybdenite is commonly concentrated by crushing and washing on tables, in a manner analogous to the concentration of graphite. Inasmuch as molybdenite flakes badly, the losses in the process are likely to be high, particularly in the slimes. Excellent separations have been made experimentally by means of the electrostatic separators, and it would be advisable to consider the application of that class of machines to the treatment of such ore.

The Elmore process of concentration by oil has been proved on a large scale to be eminently suited to the treatment of these difficult ores. At the Elmore company's testing works in London a full size two-unit plant with normal capacity of 50 tons a day is installed, where trials on bulk parcels of ore can be carried out under actual working conditions. The molybdenite concentrate produced in a recent test leaves nothing to be desired, as it is practically pure MoS_2, assaying 97.8 per cent. of the sulphide.

Uses.—In its influence upon special steels molybdenum resembles tungsten. It is used for the same purposes as tungsten, though not so extensively, the reasons being that it is more difficult to prepare in the pure state, and that its ores do not occur so plentifully. The effect it has upon steel is by no means settled, and though it is said to be from two to three times as powerful in its action as tungsten, it is not to be relied upon to the same extent, as it is more difficult to alloy with steel, because of its liability to oxidize and volatilize.

Metallurgy[1].—Molybdenum, when pure, is a silver-white metal of a high fusion point, having a specific gravity of about 9. It is as malleable as iron, and, like the latter, easily takes up carbon by cementation. In 1898 some chemical works commenced the manufacture of the metal and its ferro-alloy, and a number of experiments were made by steel-makers with variable results, no doubt largely owing to the fact that the material then at their disposal was unreliable, often containing as much as 3 per cent. of sulphur, besides several per cent. of carbon and oxygen.

Moissan first made exhaustive experiments in the electric furnace with this material. He prepared molybdic acid by heating the ammonium salt and then reduced the former by mixing it with 10 per cent. of carbon. The resulting impure metal, containing a large percentage of carbon, was again heated with molybdic acid at a temperature below fusion, thus yielding pure metal.

Later, Guichard found that by heating pure molybdenite by itself in an

[1]O. J. Steinhart, "Notes on Metals and their Ferro-Alloys used in the manufacture of Alloy Steels." *Transactions* of the Institution of Mining and Metallurgy, XV (1905-06)

electric carbon tube furnace an alloy of the following composition could be made: Mo, 91.5; Fe, 2.1; total carbon, 6.6-7.2 per cent. The ore he employed for this purpose was very pure: Mo, 60; S, 39; Fe, 0.75; Si, 0.40 per cent.

Sternberg and Deutsch, in 1892, reduced calcium molybdate by carbon, and from the reduced mass dissolved out the lime with hydrochloric acid; however, the metal still contained 3 per cent. of carbon. It was then selling for about 72c. per ℔.

According to some manufacturers wulfenite is the best raw material for the preparation of suitable metal. It is first melted with soda ash and carbon, resulting in the formation of metallic lead and a fused mass of molybdate of soda from which molybdic acid is separated. The latter is then reduced with carbon in crucibles very much like tungsten, yielding a gray metallic powder of about 95 per cent. purity with 2 to 3 per cent. carbon and a considerable quantity of undecomposed oxides, by no means an ideal product.

The working of molybdenite into MoO_3 is a difficult operation; it is effected either by a volatilizing roast, or by a fusion with sodium nitrate, and further treatment for the oxide. Some French works now make ferromolybdenum in the electric furnace, obtaining a high carbon product—Mo, 87.5; Fe, 6.4; C, 6.3 per cent.; or a low carbon alloy—Mo, 75.8; C, below 2; S, 0.5 per cent., with iron making up the total. The Thermit company now makes a 98 to 99-per cent. metal, with iron as its only impurity; it also makes carbon-free alloys of molybdenum with chromium or nickel. The present price of molybdenum, in alloy or pure metal, is $1.20 to $1.44 per ℔.

MONAZITE.

Monazite is a phosphate of cerium, in which a part of the cerium is replaced by lanthanum and didymium. The thorium oxide, which is used in the manufacture of incandescent mantles, and which gives the monazite its value, is not an essential constituent of the monazite. This thorium oxide appears to be of the nature of a mechanical admixture, and the amount of it present in a sand may, therefore, vary within wide limits. The commercial value of the monazite depends wholly, however, on the amount of thoria.

The regions in which workable deposits of monazite sand are found are few in number and small in extent. As far as present knowledge goes, such deposits occur only in North and South Carolina, on the Brazilian coast, in Norway, on the Senarka river in the Urals, Russia, in Ceylon, Queensland, and in Tringganu (one of the independent Malay states).

Monazite is of 4.9 to 5.3 sp. grav., and 5 to 5.5 in the scale of hardness. It contains from 1.2 to 14.2 per cent. of thoria, but commercially seldom averages much higher than 5 per cent. It is found principally in sands, but the ultimate source is traced to certain granites, syenites and gneisses. Large crystals (monoclinic) are found in pegmatite veins in these rocks. The color varies from yellow to brown, transparent or clouded with reddish brown decomposition products. The luster is vitreous to resinous. It is liable to be confused with yellow grains of quartz, or of epidote, zircon, and sphene. It may be readily distinguished, however, by inferior hardness, high specific gravity and the fact that it is soluble in sulphuric acid and yields the phosphate reaction with ammonium molybdate.

Monazite Mining in the United States.

The monazite region of North and South Carolina comprises 1600 to 2000 sq. miles and lies in the counties of Burke, McDowell, Rutherford, Cleveland and Polk of North Carolina, and extends south to the counties of Spartanburg and Greenville in South Carolina. The most important deposits are on the banks of the Silver river, the North and South Muddy rivers, the Henry and James forks of the Catawba river and on Broad river.

The monazite is found in the sand carried down by these rivers, and also on the bed rock of the rivers. The thickness of the sand deposits is from 1 to 2 ft. Occasionally, though rarely, the thickness is as much as 12 ft. The proportion of monazite in the suface sands is small, varying between

traces and 0.5 per cent. In 1896 the upper deposits were worked out, since which the lower strata have been worked.

Many of the heavier minerals, such as zircon, rutile, brookite, menaccanite and garnet, whose specific gravities are very near that of monazite, are but incompletely removed in the washing over riffles and through sluice-boxes, to which the raw sand is subjected. For this reason, pure monazite sand for the market cannot be secured, and a sand which contains from 60 to 75 per cent. monazite is called good quality.

Frequently the sand is subjected to further concentration in the following fashion: The sand is caused to run out of a narrow tube through a hight of 1 to 1.5 m. upon a level platform, or upon a table. As soon as a heap has built itself up, the lighter minerals, together with the finest grains of monazite, roll toward the edge of the table and are brushed off with a broom. The loss of monazite in these operations is considerable, the process is tedious, and at best the finished product carries only 65 to 75 per cent. of monazite. This roughly concentrated sand is purchased by the milling companies, who by magnetic separation (Wetherill machines) enrich the product to 95–98 per cent. monazite (about 5 per cent. thoria).

The milling companies, or producers of cleaned monazite, in the United States are the Carolinas Monazite Company, of Shelby, the German Monazite Company, of Oakspring, and the Incandescent Light and Chemical Company, of Carpenter's Knob, N. C. A recent organization is the Eureka Monazite Company, of Greenville, S. C. The supply is very closely controlled by these producers. In 1905 operations were conducted by the British Monazite Company, an organization in the interest of gas lighting concerns in Great Britain, which aimed to secure an independent supply of the mineral. However, the property which it acquired appeared upon exploitation to be of no value, and the attempt resulted disastrously.

There was a large increase in the production of monazite sand in the United States in 1905. The statistics for the last six years, as reported by the U. S. Geological Survey, are given in the following table:

Year.	Pounds.	Value.	Per pound.
1900	908,000	$48,805	$0.054
1901	748,736	59,262	0.079
1902	802,000	64,160	0.080
1903	862,000	64,630	0.075
1904	745,999	85,038	0.114
1905	1,352,418	163,908	0.121

Monazite in Foreign Countries.

Brazil.—The most important of the deposits of Brazilian monazite occur in the dunes of the extreme southern coast of the State of Bahia.

The village of Prado is the center of the industry. The sands here are continually subjected to the churning action of the waves and the shifting action of the tides. As a result, where one day there may be a place rich in monazite, its position on the next day may be quite different, or it may be out of reach altogether. This fact has placed many difficulties in the way of the prospector trying to establish good claims.

Further south, in Rio de Janeiro, monazite is found in the bed of the Parahyba river. A concentrating plant for treating the sand of this river has been erected at Lage, three miles from Sapucai.[1] This is the fourth attempt in Brazil to treat monazite sand mechanically on a large scale. The proprietor expects to be able to treat a sand containing only 2 per cent. monazite and recover a product, worth $135 per ton, at the rate of 50 tons per month. The installation consists of a trommel, which separates from the gravel the finer particles of sand in which all the monazite is contained. This material is passed over eight Wilfley tables to separate the lighter associated minerals, as quartz, etc., and then through seven magnetic separators to separate the iron minerals.

The exports of monazite sand from Brazil were 1205 metric tons in 1902; 3299 tons in 1903; 4860 tons in 1904; and 4437 tons in 1905. The average export value in 1904 was 448 milreis per ton (11.06c. per ℔); and in 1905 it fell to 347 milreis per ton (8.57c. per ℔.)

Ceylon.[2]—The analysis of two specimens of uraninite and monazite, found in the refuse from gem washings near Balangoda, Ceylon, are interesting as suggesting a possible commercial source of thorium minerals. The first specimen occurred in black cubical crystals, the specific gravity of which was 9.32. Its analysis was as follows: Thorium oxide, 76.22; cerium, lanthanum and didymium oxide, 8.04; zirconium oxide, trace; uranium oxide, 12.33; ferric oxide, 0.35; lead oxide, 2.87; silica, 0.12 per cent.

This mineral is clearly not uraninite nor pitchblende, as the percentage of uranium is only about 12, while thoria is present in amounts of over 75 per cent., higher than that contained in any mineral hitherto known. This mineral is new, and the name thorianite has been given to it. It has since been found that radium is present in the mineral.

The second mineral consisted of fragments with a deep brown color which exhibited, when fractured, a purple brown with a resinous luster. Its specific gravity was 4.98. It was chiefly composed of thorium silicate, and therefore is not monazite but thorite.

Malay Peninsula.[3]—Monazite exists in connection with cassiterite in Tringganu, an independent native State on the east coast of the Malay Peninsula. On one concession, worked for tin by Chinese, one-half of the

[1] *Brazilian Review*, Rio de Janeiro, Sept. 19, 1905.
[2] W. R. Dunstan, Ceylon Mineral Survey, III.
[3] C. G. Warnford Lock, *Bulletin* 17, Institution of Mining and Metallurgy.

material collected in the sluice-boxes was found to be monazite sand.

Queensland.—The occurrence of monazite in the beach sands of Queensland, near the mouth of the Tweed river, has been known for some time. In the Walsh and Tinaroo fields it has also been found in tin-bearing gravel, but in greater abundance in gravels that are washed for scheelite; from this, monazite is separated with difficulty, being often confused with it by the miners.

At two localities monazite has been found in its original habitat, associated with wolframite, molybdenite, scheelite, cassiterite and arsenopyrite. It occurs in pure crystalline masses, sometimes several pounds in weight, and also in small grains, both forms being irregularly disseminated in a greisen. An analysis of a clean sample of monazite showed 66 per cent. of the rare earths; a commercial sample, without cleaning or concentrating, yielded 2.6 per cent. thoria and 56.1 per cent. of the rare earths.

Commercial Conditions.

The monazite of commerce is divided into three classes:

1. Brazilian monazite—fine, amber colored, rounded grains.

2. Carolina monazite of Cleveland county—well formed, sharp-edged, yellow crystals mixed with the component minerals of the country rock of the region, such as chromite, ilmenite, garnet and zircon.

3. Carolina monazite from the northeastern spur of the Blue Mountains —dark brown crystals of about pea size.

The particles of Brazilian monazite are so small that they are readily soluble in concentrated sulphuric acid, but the Carolina varieties must first be ground in order to render them soluble.

The monazite of the Carolinas is obtained chiefly by small producers, who dispose of the roughly concentrated sand to the companies which operate the mills. The latter have an actual (or tacit) agreement to pay the same price. Early in 1905 there was a flurry in the market, and, for a while, raw monazite reached a price of from 15 to 20c. per ℔.; but it soon returned to about 5c. per ℔.; the different interests do not bid against each other.

The monazite industry of the Carolinas has recently been disturbed by the invasion of the American market by the syndicate of German thoria producers, which controls the Brazilian supply of monazite. In the spring of 1906, the price of thorium nitrate in the United States was cut nearly 50 per cent. It is anticipated that this will freeze out some of the smaller concerns, and it has already seriously affected the mining operations in the Carolinas. Monazite sand, containing 5 per cent. of thoria, can hardly be delivered in New York even by the largest producers at less than 14 or 15c. a pound, but at present it will hardly bring that.

The exploitation of the Brazilian sands was originally undertaken by John Gordon, of the firm of Edward Johnston, Son & Co., of London and Rio de Janeiro. The sand was considered worthless and Gordon was able to ship it as ballast to Hamburg for as little as $10 to $15 per ton. He sold it at a good profit, the lowest price for sand carrying 5 per cent. thorium oxide being $95 to $120 per ton.

Gordon aimed at a monopoly of the export of Brazilian monazite, but eventually he was obliged to enter into an agreement with a combination of manufacturers of thorium products. By the terms of the agreement Gordon was to sell his sand (exclusively to members of the combination) for $150 per ton and was to guarantee at least 5 per cent. thorium oxide. He also received a share in the net profits from the sale of the thorium nitrate.

In 1903 the legislative assembly of Brazil discovered that according to an old statute the ownership of the sands along the coast belonged to the Federal Government, and not to a single State or an individual, and so the exploitation of the sands by Gordon was forbidden. However, Gordon was able to lease deposits on a royalty basis.

In the summer of 1903, the Federal Government of Brazil called for bids for the privilege of working deposits on the south coast of Espirito Santo, and ultimately this was secured by the A. C. de Freytas Company, of Hamburg. This firm guaranteed a royalty of 50 per cent. of the revenue produced by the sale of the sand and also guaranteed a yearly production of 1700 tons. A new agreement was concluded between Gordon, the de Freytas company, and the German Thorium Company. By the terms of this agreement, Gordon and de Freytas supply monazite only to the firms belonging to the German Thorium Company, and moreover receive a part of the profits of the sale of thorium nitrate.[1] Their share in the profits is said to be 50 per cent. of the price received in excess of 28 marks per kilogram.

According to the estimate of a Hamburg importer, the amounts of monazite brought to Germany from Brazil since 1900 have been 1500 tons in 1901; 1500 in 1902 and 2000 in 1903.

It appears that there are fairly good opportunities to secure good monazite properties at reasonable figures, and that the German Thorium Company is having more and more difficulty in keeping up its monopolistic prices. This became evident in January, 1906, when the price for thorium nitrate, which for a long time had been 53 marks per kg., suddenly fell to 27 marks. Commercial thorium nitrate contains 47.5 per cent. of thoria.

The German Thorium Company includes Dr. O. Knoefler, of Berlin; Kunheim & Co., of Berlin; and Dr. Richard Sthamer, of Hamburg. In the United States, the Welsbach company, of Gloucester, N. J., and one

[1]C. R. Böhm, *Chemische Industrie*, Jan. 1, 1906, pp. 2-7.

other small concern make the nitrate. The Welsbach company controls the Carolinas Monazite Company. The Welsbach company is the largest producer of mantles, but there is said to be about 75 outside concerns, most of whom buy their thorium nitrate from Germany. There is an import duty of 25 per cent. *ad valorem* on the nitrate, and 6c. per lb. on monazite sand.

NICKEL AND COBALT.

By Edward K. Judd.

The United States is not a regular producer of nickel or cobalt ores; by-products containing these metals are recovered in irregular amounts by certain of the lead miners and smelters of Missouri, especially the Mine La Motte, but in recent years their production has been insignificant. Discoveries of ores reported to contain nickel or cobalt are reported, from time to time, from Idaho, Oregon, North Carolina, Virginia, Washington and Wyoming, but thus far, none of these localities has yielded either of the metals. The raw materials required for nickel and cobalt smelting are, therefore, supplied by imports of ore and matte from Canada. Imports of nickel ore and matte in 1905 resumed nearly their size previous to 1904, in which year they fell off one-half. The value per ton of the imported raw material shows a marked advance in recent years. Much of the imported nickel-bearing matte from Canada enters as copper matte, so that the above table does not account for all the nickel that is ultimately recovered from foreign sources. Both copper and nickel mattes enter free of duty, for which reason the distinction between the two is not carefully drawn.

UNITED STATES IMPORTS AND EXPORTS OF NICKEL AND COBALT.
(In pounds, and tons of 2240 lb.)

Yr.	Imports.							Exports.	
	Nickel Ore and Matte.		Nickel Alloys. (a)		Nickel Mnfrs.	Cobalt Oxide.		Nickel. (b)	
	Long Tons.	Value.	Pounds.	Value.	Value.	Pounds.	Value.	Pounds.	Value.
1896	10,589	$620,425	(c)	....	.....	27,189	$36,212	2,756,604	$606,833
1897	12,420	781,483	(c)	....	.....	24,771	34,773	4,255,558	997,391
1898	26,826	1,534,262	(c)	....	.....	33,731	49,245	5,657,618	1,359,609
1899	19,857	1,216,253	(c)	....	.....	46,791	68,847	5,004,377	1,151,923
1900	25,670	1,183,884	455,188	$139,786	.....	54,073	88,651	5,869,906	1,382,727
1901	52,111	1,637,166	635,697	209,956	$2,498	71,969	134,208	5,869,655	1,521,271
1902	14,817	1,156,372	752,630	251,149	30,128	79,984	151,115	3,228,607	925,579
1903	15,936	1,285,935	521,344	170,670	37,284	73,350	145,264	2,414,499	703,550
1904	8,548	915,470	589,555	203,071	2,950	42,352	86,925	7,519,206	2,130,933
1905	13,451	1,626,920	941,966	331,920	3,291	70,048	139,377	9,550,918	2,894,700

(a) Includes nickel, nickel oxide, and alloy of any kind in which nickel is the material of chief value, in ingots, bars and sheets. (b) Comprises domestic nickel, nickel oxide and matte. (c) Not separately enumerated; included in "Nickel Ore and Matte."

The nickel market showed nothing noteworthy during the year. Prices were a little lower than in 1904, holding throughout at 40@37c. per ℔. in large lots—down to one ton—and ranging around 60c. per ℔. for small orders.

The production of metallic nickel, oxide of nickel, etc., in the United States is entirely in the hands of the International Nickel Company, which operates the Orford Works at Constable Hook (Bayonne), N. J., and

the American Nickel Works at Camden, N. J., the former being by far the more important. At these works, nickel-copper matte from Sudbury, Ontario, is refined by the "tops and bottoms" process, which has been described in previous volumes of THE MINERAL INDUSTRY. Improvements in this process are referred to in a subsequent section of this article. The Balbach Smelting and Refining Company, at Newark, N. J., used to produce electrolytic nickel from material obtained from the International Nickel Company, but its plant was not in operation during 1905. Owing to the concentration of the nickel production of the United States in one company, we are unable to give a direct report of its production. We estimate, however, that this amounts to the equivalent of 6000 to 7000 tons of metallic nickel per annum.

An important new feature in the nickel industry toward the end of 1905 was the introduction in the market, by the International Nickel Company, of a new alloy, called "Monell metal," in honor of Mr. Ambrose Monell, president of the company, to whom the process of preparing it is due. The composition of this alloy is slightly variable, but is about as follows: Nickel, 68 to 70 per cent.; copper, 30 to 28 per cent.; iron, 2 per cent. The iron is an unavoidable impurity. The process of manufacture is a secret one. However, it may be said that the metal is not the result of simple mixing, but made by the treatment of nickeliferous copper matte whereby everything is removed except the constituent parts of the alloy and the small amount of iron above noted.

The aim of the inventor was to produce an alloy having some of the characteristics of both nickel and copper. In this he has succeeded and the metal which is now being marketed is of considerable interest. In appearance it much resembles nickel. It is capable of taking a high polish, and is not corroded any more than copper. Being homogeneous, the surface does not wear as does nickel-plate and its ductility allows of great ease of working. With these copper-like qualities are combined strength equal to steel. A tensile strength between 85,000 and 95,000 lb. enables the manufacturers to guarantee their metal at 80,000 lb. per sq. in. The elongation in 8 in. is between 32 and 40 per cent., accompanied by a reduction of area of between 50 and 60 per cent. In ingot form the metal sells for 22 to 23c. per lb. The metal may be spun, drop forged, worked hot or cold and under skillful management even castings can be made. The alloy is being manufactured into boiler-tubes, fire-boxes, sheet-metal goods of all kinds, and is suitable for anything where strength and non-corrodibility are requisite.

NICKEL AND COBALT IN THE UNITED STATES.

Missouri.—The only commercial producer of nickel and cobalt in this country, recently, has been the old Mine La Motte, to whose output of

nickel-cobalt matte reference has been made. Another lead producer in that vicinity, the North American Lead Company, of Fredericktown, is now seeking to treat a nickeliferous ore which it has encountered in extensive quantities. This company's lead mining operations now yield, as a by-product, 50 tons per day of sulphides carrying 5 per cent. of copper and 3 per cent. each of nickel and cobalt, and the management is planning the erection of a furnace for smelting this.

Oregon.—The Standard Consolidated Mines Company, operating in eastern Oregon, owns a series of veins, 3 to 14 ft. wide, which contain gold-silver ore carrying cobalt, a little nickel and a high percentage of arsenic. A mill run on 16 tons of average ore yielded gold, $250; copper, $34; cobalt, 420 ℔. The ore consists of smaltite, arsenopyrite, and pyrite and chalcopyrite. A picked specimen of the ore showed: Ni, 0.57; Co, 9.91; As, 42.66; Fe, 14.93; CaO, 2.16 per cent.; Ag, 5.22 oz. and Au 1.62 oz. per ton.

An Elspass mill was installed, early in 1906, with a capacity of 50 tons per day. The ore is crushed and run through a 10-mesh screen and the pulp passed over a 40-mesh shaking screen, the oversize going to a Wilfley table and the undersize to hydraulic classifiers. From the classifiers the coarse product goes to a Cammett table and the fine sands to a Card table, while the slimes pass to another Wilfley table, where two products are cut out, cobalt concentrate and copper concentrate. The concentrates from the three tables are re-concentrated on the Wilfley table to separate cobalt and copper concentrates. It is estimated that the Standard mine with its 50-ton mill will produce from 1000 to 2000 ℔. of cobalt per day. One of the largest buyers of cobalt in the country has had its representative at the mine with a view of purchasing the entire cobalt product.

Nickel and Cobalt Mining in Foreign Countries.

Canada.—The nickel ores of Sudbury and the more recently discovered silver-cobalt ores of Timiskaming, 90 miles to the northeast, have attracted

PRODUCTION, EXPORTS AND IMPORTS OF NICKEL IN CANADA. (a)

Year.	Production.		Exports.		Imports.
	Pounds. (b)	Value. (c)	Pounds (d)	Value. (e)	
1900	7,080,227	$3,327,707	13,493,239	$1,040,498	$6,988
1901	9,189,047	4,594,523	9,537,558	958,365	12,029
1902	10,693,410	5,025,903	3,883,264	834,513	15,448
1903	12,505,510	5,002,204	9,032,554	878,159	26,177
1904	10,547,883	4,219,153	14,229,973	1,237,307	16 330
1905	18,876,315	7,550,526	11,970,557	1,185,056	19,076

(a) Statistics for production and imports cover calendar years, and are taken from the Annual Reports of the Geological Survey of Canada. Figures for exports cover the fiscal years ending June 30, and are taken from the Statistical Year Book. (b) Pounds of metallic nickel contained in copper and nickel matte exported (c) On the basis of refined nickel at New York, from the *Engineering and Mining Journal* average annual quotations. (d) Pounds of nickel contained in ore, matte or speiss. (e) Spot value, to the producer, of the exported material; the variety of stages at which the material is shipped, as well as the different periods of time covered, lead to the apparent discrepancy in value when it is known that practically the entire production is exported.

wide attention. The geological features of the former have been described by A. E. Barlow[1] for the Canadian Geological Survey, and of the latter by W. G. Miller,[2] for the Ontario Bureau of Mines.

Of the total export in the fiscal year 1905, Great Britain took 802 tons and the United States took 5183 tons, the figures in the table representing the fine nickel contents of the ore, matte and speiss exported.

The condition of the nickel mining industry in Ontario may be represented thus:

Schedule.	1900.	1901.	1902.	1903.	1904.	1905.
Ore raised	216,695	326,945	269,538	152,940	203,388	277,766
Ore smelted	211,969	270,380	233,388	220,937	102,844	251,421
Per cent. nickel	1.67	....	2.55	3.16	4.58	(b)
Per cent. copper	1.59	....	1.78	1.81	2.41	(b)
Ordinary matte	23,336	29,588	24,691	30,416	19,123	17,388
Bessemerized	112	15,546	13,332	14,419	6,926	
Nickel content	3,540	4,441	5,945	6,998	4,743	9,438
Copper content	3,364	4,197	4,066	4,005	2,163	4,386
Value of nickel (a)	$756,626	$1,859,970	$2,210,961	$2,499,068	$1,516,747	$4,019,814
Value of copper (a)	319,681	589,080	616,763	583,646	297,126	
Wages paid	728,946	1,045,889	835,050	746,147	570,901	(b)
Men employed	1,144	2,284	1,445	1,277	1,063	(b)

NOTE.—The quantities reported in 1901, 1902 and 1903 under "bessemerized matte" include both bessemerized matte and high-grade matte, the former being the product of the Mond Nickel Company's works and the latter of the Ontario Smelting Works, which re-treat the low-grade matte produced by the Canadian Copper Company. (a) Value based on nickel and copper in matte and not on refined metals. (b) Not available.

The shipments of ore from the Cobalt mining camp for 1905 aggregated 2144 tons from 17 shipping mines. The metal contents were as follows: Silver, 2,441,421 oz., valued at $1,355,306; cobalt, 118 tons, valued at $100,000; nickel, 75 tons, valued at $10,525; and arsenic, 549 tons, valued at $2693—being an aggregate value of $1,448,524 received by shippers. These figures, so far as relates to the latter three items, were considerably reduced by the refusal of buyers during the latter part of the year to make any allowance for the nickel, cobalt or arsenic contents of the ore on account of its refractory character. This has led to several schemes for the erection of local smelting works, and it is possible that some attempts in that direction will be carried out in 1906.

The average contents of the shipments during the first six months of 1905 were: Arsenic, 31 per cent.; nickel, 3.6 per cent.; cobalt, 7.3 per cent., and silver, 1,257 oz. per ton.

The price offered for cobalt fell from 65c. to 35c. per lb., and while formerly 12c. per lb. was paid for the nickel and ½c. per lb. for the arsenic contents of the ore, nothing is now allowed for these constituents. The price paid for the silver is 90 per cent. of the current rate for fine silver.

On Aug. 28, 1905, the Ontario Government withdrew all lands in the townships of Coleman, Bucke, Lorrain and Hudson, comprising the silver-cobalt area, from sale or lease under the Mining Act. This step was taken

[1]Geological Survey of Canada: "Report on the Origin, Geological Relations and Composition of the Nickel and Copper Deposits of the Sudbury Mining District in Ontario," by Alfred Ernest Barlow, Ottawa, Canada; Public Printer. 236 pp., with maps and illustrations. Price, 25 cents.

[2] "The Cobalt-Nickel Arsenides and Silver Deposits of Timiskaming." Part II of the 1905 report. By Willet G Miller. 66 pp.; illustrated. Toronto, Ontario, 1905: Bureau of Mines.

to enable the Government to adopt new regulations before any further mining rights were granted, with the object of making the mines yield a larger revenue to the province by imposing a royalty or percentage upon the profits from mining enterprises and by reducing the area of claims. The order-in-council provided that existing rights of applicants are not to be interfered with.

A subsequent order-in-council reduced the area which will in future be granted to applicants for mining locations in Coleman township from 40 to 20 acres. No reduction was made in the area of locations in other townships than Coleman. Except at Cobalt lake, Kerr lake and the Gillies timber limit, which are not open to prospectors, locations were subsequently permitted, subject to any amendments which the legislature might later make in the mining laws. There are now 25 working properties in the Cobalt mining area.

New Caledonia.—The nickel mining industry of this island appears to have recovered its former activity, in spite of a constantly diminished price paid for the ore by the nickel trusts. The shipments of nickel and of cobalt ores for the last seven years have been as follows:

SHIPMENTS OF NICKEL AND COBALT ORES FROM NEW CALEDONIA. (a)
(In metric tons.)

	1899.	1900.	1901.	1902.	1903.	1904.	1905.
Nickel ore	103,908	100,319	133,676	129,653	77,360	98,655	125,289
Cobalt ore	3,200	2,400	3,110	7,512	8,292	8,961	7,919

(a) Reported by *Le Bulletin du Commerce*, Noumea.

The exports of nickel ore in 1905 were distributed thus: 53,701 tons to England, 34,424 tons to France, 26,100 to Holland and the rest to Germany and Australia. The cobalt mines are still sustained by favorable contracts made in 1904; only the most favored are still working. Of the cobalt ore exported in 1905, England took 3352 tons; France, 2238 tons; Australia, 1792 tons, and Germany the remainder.

A new scale of royalties promulgated in 1905 levies 1.25 francs ($0.24) per ton on nickel ore, and 2 francs ($0.39) per ton on cobalt ore. The average price for 7 per cent. nickel ore is now 0.60 francs (11.6c.) per kg. of contained metal; sales of cobalt ore occur but seldom, but an average price is 125 francs ($24.13) per ton of 4 per cent. ore.

Progress in the Metallurgy of Nickel.

Nickel-Copper Matte Refining.—N. V. Hybinette, Westfield, N. J., describes[1] a process wherein the matte is roasted to convert the metals into oxides, and then is leached with weak sulphuric acid, which extracts prin-

[1] U. S. patent (Orford Copper Co.) No. 805,555, Nov. 28, 1905.

cipally copper. The residue is heated with sulphuric acid at least to a temperature at which hydrous sulphates do not exist, and then is again leached with weak sulphuric acid to extract copper. The residue is then heated with hydrochloric acid to a temperature sufficiently high for partial decomposition of the anhydrous chlorides and again leached with weak acid, the heatings being repeated, if necessary, in order to obtain a residue of nickel oxide suitable for refining by ordinary means.

The same inventor describes[1] a process for the separation of nickel from copper, which consists in making an alloy of the two metals the anode in an electrolyte consisting of a solution of nickel sulphate and a "weak acid," contained in a cell divided into anode and cathode compartments by a porous diaphragm. The process is so conducted that the solution in the cathode compartment contains only nickel, while that in the anode compartment contains both nickel and copper, and the passage of copper through the diaphragm is prevented by causing the electrolyte to circulate from the cathode to the anode, and by maintaining a pressure in the cathode compartment. The anode solution is regenerated, i. e., freed from copper, by "cementation" on nickel or a nickel-copper alloy refined at least to the point at which carbon, silicon and sulphur are practically eliminated, and is then returned to the cathode compartment.

Nickel-Copper Separation.—Ambrose Monell, of New York, has described an improvement in the method of separating nickel and copper sulphides, as practised at the Orford works.[2] The following, which is taken from the specification of the patent, reviews the previous state of the art and outlines the improvement:

In the reduction of ores containing nickel and copper where a matte is produced containing sulphides of nickel, copper and iron, a process has been devised in which a separation of the nickel sulphides is effected by the use of sodium sulphide, advantage being taken of its power of dissolving the sulphides of copper and iron freely and forming a solution of less specific gravity than the nickel sulphide. The matte mixed with coke and sulphate of sodium has been charged into a cupola-furnace. When this charge is smelted, the sodium sulphate is reduced by the coke to sodium sulphide and, forming a solution with part of the copper sulphide and iron sulphide, flows with the undissolved and melted sulphides of nickel, copper, etc., through the tap-hole, which is kept constantly open, into molds, where the molten constituents separate in accordance with their specific gravity, the sodium sulphide containing the dissolved copper and iron sulphides floating on the surface and the undissolved sulphides settling to the bottom. When the contents of the mold have solidified, the parts are separated by fracture, and the tops containing the copper and iron are recharged into

[1] U. S. patent No. 805,969, Nov. 28, 1905.
[2] U. S. patent No. 802,012, Oct. 17, 1905.

a smelting-furnace, where the sodium sulphide is fluxed off in an iron slag, being then lost. The bottoms contain most of the nickel sulphide of the original matte; but owing to the imperfection of the separation they also contain so much copper sulphide and iron sulphide that it is necessary to resmelt them with fresh additions of coke and sodium sulphate, and thus to repeat the smelting and separation to the fourth or fifth time before the bottoms are brought to sufficient degree of freedom from iron and copper to enable the resultant nickel sulphide to be economically subjected to the succeeding steps of the refining process. The process as thus carried on is slow and wasteful, and because of the cost of materials and the amount of labor and handling required adds greatly to the expense of the nickel or nickel oxide which are the final products. These difficulties are overcome by the following process:

Instead of smelting the compound matte, as heretofore, in a cupola-furnace and running the product continuously into molds, the matte is so smelted that it will remain in a molten state subject to the high temperature of a furnace for a considerable period of time, during which the copper and iron sulphides are thoroughly dissolved by the sodium sulphides, and in one melting a good separation can be effected; and by two such treatments results are obtained equal or superior to the results of the four or five meltings which have been employed heretofore. The smelting-furnace employed for this purpose is an open hearth reverberatory, lined with magnesite brick; silica linings are quickly destroyed by fluxing with the sodium sulphide. A charge of nickel-copper-iron matte, either solid or molten, is mixed with coke and sodium sulphate, the latter being preferably present in the proportion of 60 per cent. of the weight of the matte and the coke in the proportion of 15 per cent. of the matte. The sulphate is preferably added in the form of commercial niter-cake. Where, for example, a 50-ton charge of matte is treated containing, say, 45 per cent. of nickel sulphide and 35 per cent. of copper sulphide, it is melted in the furnace and retained subject to the heat for four or five hours after fusion has occurred, during which time it is preferably "poled," i. e., treated by immersing beneath its surface poles of green wood, which facilitates and renders more thorough the solution of the sulphides to be removed. Nearly complete solution of the copper and iron sulphides in the sodium sulphides reduced from the niter-cake is thus effected, and the molten charge may be tapped from the furnace and allowed to separate in molds; but to get the best results, the different strata are tapped from the furnace separately, tapping first the solution of copper and iron sulphides floating on the surface of the bath, and finally tapping the undissolved nickel sulphide, or the order of tapping may be reversed, the lower stratum of nickel sulphide being removed first. The great proportion of the iron and copper is thus separated, the nickel sulphide obtained being nearly pure. Where

greater purity is desired, the nickel sulphide may be recharged into the furnace and treated again in like manner.

Robert R. Maffett, of Bayonne, N. J., describes[1] a new method of treating copper tops in the refining of nickel-copper matte. He takes the copper-iron tops from the nickel cupolas and piles them, preferably in a heap of about 3 ft. deep, in the open, where the action of the weather oxidizes the sodium sulphide to sodium sulphate. These tops are allowed to weather about eight weeks, the greater part of the sodium sulphide in the tops being oxidized to sodium sulphate, which is easily separated from the copper sulphide and nickel sulphide contained in the tops. As soon as the tops which have been treated in this manner are smelted, the sodium sulphate rises to the top of the charge and is drawn off into pots, when it is allowed to cool and is returned to the nickel cupolas, where it is used in place of refined saltcake. The conversion of the sodium sulphide to sodium sulphate is completed by oxidation, which is effected, preferably, by an air-blast, and the oxidation is continued until all or nearly all of the sodium sulphide is converted into sulphate, which in a coal-fired reverberatory furnace containing, say, 25 tons of the tops may last 12 hours after the fusing of the charge; but the time will vary with the heating capacity of the furnace and the rapidity of the oxidation. When the conversion of the sodium compound to sodium sulphate has been completed, the molten contents of the furnace may be withdrawn together into pots or molds, when they will separate by gravity with a clean sharply defined separation into a floating stratum of sodium sulphate, a bottom stratum of copper sulphide carrying a small percentage of nickel sulphide, and an intermediate stratum of iron silicate which may carry about 3 per cent. of copper and 1.5 per cent. of nickel in the form of oxides. While the stratified and congealed mass is cooling and before it has entirely chilled, the layers can readily be separated from each other by tapping with a hammer. The top layer, consisting of sodium sulphate, carrying a little copper and nickel, can be used again in the separation of matte, the iron layer can be charged directly into a smelting-furnace, and the bottom layer of copper sulphide, being nearly free from iron, can be refined in a reverberatory furnace without need of previous calcination.

Instead of tapping the molten products of the separating-furnace together and allowing them to separate in pots or molds, as described above, advantage may be taken of their stratification while in the furnace and the layers can be tapped separately into different vessels, since in this way a more complete separation and purer products can be obtained. By skimming or tapping off the floating layer of sodium sulphate and iron silicate and leaving the copper sulphide remaining in the furnace, the further refining of the copper may be, if desired, directly continued in the same furnace.

[1]U. S. Patent No. 802,148, Oct. 17, 1905.

Casting Nickel Anodes[1].—Owing to the difficulty of casting pure nickel into anodes as required for nickel plating out of nickel-ammonium sulphate solution, the earlier workers attempted to use granulated nickel for anodes, suspending it in bags and perforated boxes. The removal of the accumulated residue, in order to maintain a low resistance, was an insurmountable difficulty of this method. It was then found that nickel alloyed with iron and tin not only could be readily cast into any desired shape, but that it was even more suitable for anodes than pure nickel. An anode of the pure metal collects hydrogen on its surface, to the detriment of its solubility; an alloyed anode, on the other hand, sets up local action whereby the hydrogen is dispelled while the impurities merely form a slime in the tank.

Nickel anodes are now made in the following grades: 80-82; 82-84; 85-87; 90-92; and 92-97 per cent. nickel, the variation being confined to the iron contents, while the tin remains constant. Three anodes, of the different grades, showed the following composition:

Grade.	90%	85%	80%	Grade.	90%	85%	80%
Nickel	90.13	81.11	77.72	Silicon	1.12	1.56	2.17
Iron	4.67	12.15	15.16	Copper	.09	.19	.11
Tin	3.20	4.01	3.89	Carbon	.79	.98	.95

To form the alloy, about 100 ℔. of nickel, in buttons, grains, cubes, or shot, is mixed with the proper amount of iron, preferably old files (containing the maximum proportion of combined carbon) and fused in a No. 40 graphite crucible with 5 ℔. of a flux composed of lime, 3 parts, and fluorspar, 1 part, and a little charcoal. The fusion is cast into ingots and re-melted; the tin is added to the second fusion just before the final pouring.

The molds are formed in iron flasks with a fine-grained molding sand and are thoroughly dried at a high heat before casting. The mold should be gated at the bottom of the pattern, so that the metal may be purest at the top of the mold, where it is suspended, so as to avoid excessive corrosion of the anode at the bath surface. The casting is best cleaned by a sand blast, which leaves the surface rough and well adapted as an anode. The holes for supporting the anode are then drilled through the lugs on the casting.

[1]Edwin S. Sperry in *The Brass World*, 1905, p. 327

PETROLEUM.

By F. W. Parsons.

The development of the petroleum industry during 1905 was rather disappointing for those who had expected the production again to show the encouraging increase recorded in the preceding year, which surpassed 1903 by about 3,000,000 tons. It will undoubtedly be a surprise to many to find that, in spite of the greatly increased activity which characterized the operations in many of the important fields, the total production of 1905 falls considerably below that of 1904. This falling off, however, was due almost entirely to the troubles in the Caucasus, which greatly diminished the Russian production. The United States showed a substantial increase.

Each year the petroleum market has rapidly extended, and the supply has kept even pace with this increasing demand; but 1905 marks a period of halting in the yield, which should have materially increased the price of crude oil, and not doing so, the search for new fields, and the extension of old ones, was not carried on with the stimulated activity that has characterized other periods. Had Russia continued its normal production in 1905, the world's yield would have shown almost as favorable an increase as did the preceding year; consequently the circumstances which brought about this year's decrease are not of a nature to cause the least apprehension as to the future of the industry, and if nothing unusual happens to prevent, it is fairly certain that 1906 will show a resumption of activity equal to the most favorable of preceding years.

The production of the respective petroleum producing countries for the last four years, according to statistics published in the *Petroleum Review*, has been as follows, in metric tons:

PETROLEUM OUTPUT OF THE WORLD.

	1902	1903	1904	1905
	Tons.	Tons.	Tons.	Tons.
United States	10,980,000	12,557,000	15,000,000	17,000,000
Russia	10,950,000	10,320,000	10,600,000	6,500,000
Sumatra, Java and Borneo	732,000	830,000	1,000,000	1,200,000
Galicia	576,000	713,800	827,000	800,000
Rumania	310,000	384,300	455,000	568,000
India	209,000	325,400	404,000	465,000
Other countries	270,000	250,000	250,000	350,000
Total	24,027,000	25,380,000	28,577,900	26,883,000

From the above figures, it will be noted that the United States not only maintains its lead as the greatest producing country in the world, but

continues to advance at a rate which far surpasses that of any other country. Its production in 1905 was 17,000,000 tons, or 2,000,000 tons in excess of the previous year.

A review of the American fields brings into prominence the fact that while remarkably rapid headway has been made in the development of those west of the Mississippi, the regions of Pennsylvania, West Virginia, Kentucky and Tennessee—or, in other words, the fields supplying high grades of oil—are constantly declining in their output. The decline, however, is counteracted by the results which have attended the developments in the newer districts of the West.

The mid-continental field in 1905 more than doubled the yield of 1904, while in California the increase in production amounted to nearly 1,000,000 tons. Texas also produced, roughly, one and a half million tons more than the year preceding, but the production in this State fell off greatly during the latter part of the year, due to the rapid decline in the yield from the Batson and Humble districts.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES.
(In barrels of 42 gal.)

Field.	1900	1901	1902	1903	1904	1905
California (a)	4,250,000	8,786,330	13,973,500	24,337,828	28,476,025	35,671,000
Colorado	525,000	460,520	396,901	483,925	(b) 501,763	(e) 550,000
Gulf: Texas	800,000	4,393,660	18,083,658	17,955,572	21,672,111	30,354,263
Gulf: Louisiana	....	....	548,617	917,771	6,611,419	9,672,015
Lima: Indiana	4,329,950	5,757,086	7,535,561	9,177,122	10,744,849	22,102,108
Lima: Ohio	16,407,704	16,176,293	15,877,730	14,893,853	13,350,060	
Mid-Continental (c)	65,000	179,150	359,123	1,157,110	5,617,527	12,000,000
Ken.—Tennessee	(f)	(f)	(f)	(f)	998,284	(e) 1,200,000
Pennsylvania (d)	35,540,965	33,618,180	32,018,787	29,897,815	(b)30,410,183	28,324,324
Wyoming	7,200	5,400	6,253	8,960	11,542	(e) 12,500
Others	30,000	2,585	957	3,000	2.572	(e) 3,000
Total	61,955,819	69,379,204	87,801,087	98,832,956	118,396,335	139,889,210

(a) Reported by the California Producers' Association. (b) Statistics of the U. S. Geological Survey. (c) Kansas, Indian Territory and Oklahoma. (d) Pennsylvania, New York, West Virginia, Eastern Ohio, and, until 1904, Kentucky and Tennessee. (e) Estimated. (f) Included in Pennsylvania.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (In gallons.)
(1=1000 in quantities and values) (a)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffin.		Residuum. (b)		Totals.	
1896	118,133	$6,032	13,641	$1,123	758,076	$49,704	51,705	$6,771	521	$28	942,076	$63,658
1897	121,864	5,044	13,704	1,020	804,446	46,876	52,659	6,732	12,247	335	1,004,920	60,007
1898	120,436	5,016	17,258	1,071	764,823	38,895	65,526	7,626	30,436	815	998,479	53,423
1899	117,690	5,958	18,210	1,597	733,382	49,172	71,116	8,658	21,609	658	962,007	66,043
1900	133,161	7,341	18,570	1,681	739,163	54,693	71,211	9,933	19,750	845	986,855	74,493
1901	127,008	6,038	21,685	1,742	827,479	53,491	75,306	10,260	27,596	1,255	1,079,059	72,786
1902	145,234	6,331	19,683	1,393	778,801	49,079	82,200	10,872	38,316	922	1,064,234	68,597
1903	126,512	6,782	12,973	1,519	691,837	51,356	95,622	12,690	9,753	282	936,697	72,629
1904	111,176	6,351	24,989	2,322	761,358	58,384	89,738	12,389	34,904	1,174	1,022,165	80,620
1905	126,185	6,086	28,420	2,215	881,450	54,901	113,730	14,312	70,728	2,128	1,220,513	79,641

(a) In addition to the above, the following quantities of paraffin and paraffin wax were exported: 1896, 112,517 lb. ($4,563); 1897, 136,069 lb. ($5,284); 1898, 166,317 lb. ($6,363); 1899, 181,861 lb. ($7,650); 1900, 157,108 lb. ($8,186); 1901, 151,695 lb. ($7,960); 1902, 175,268 lb. ($8,398); 1903, 204,120 lb. ($9,596); 1904, 174,582 lb. ($8,273); 1905, 160,836 lb. ($7,873). (b) Reported in barrels of 42 gallons.

The part that America is playing in the world's petroleum output is readily recognized when we note that in 1904 it produced 52 per cent., and in 1905, 65 per cent. of the world's yield.

It is especially worthy of note that the Dutch Indies—including Sumatra, Java and Borneo—rank third on the list of producing countries. It is in these regions that we may in the future expect to witness remarkable developments, for they only need increased refinery capacity and extensions of the various plants to increase rapidly in production.

With Canada now producing considerable oil of fair quality, and active exploitation going on or about to commence, in Peru, Turkestan, New Zealand, Argentina, Africa and China, it is very probable that the future may reveal further important fields to help swell the production of this highly necessary substance, which has become a commercial factor in every quarter of the globe.

Review by States.

California.—The California oilfields showed great activity in 1905, the increased output being accompanied by an extension of the marketing facilities. California brought in the largest producer of the year, this having started at 10,000 bbl. per day; at the end of the year it was still producing 4000 bbl. The export demand increased to an extent which gives the producer encouragement to expect better prices for his product.

The Kern river field is the most important yet developed in California, and promises to be one of the greatest fields in the world. This district alone has paid over $2,000,000 in dividends during the last three years.

PRODUCTION OF CRUDE OIL IN CALIFORNIA.
(Reported by the Petroleum Miners' Association).

District.	1902	1903	1904	1905
Coalinga	500,750	2,138,058	5,114,000	8,869,000
Sta. Maria and Lompoc	116,500	208,890	670,500	5,300,000
Kern River	8,872,115	16,342,100	17,500,000	14,000,000
Los Angeles	1,047,300	793,765	1,200,000	3,000,000
Sunset	144,200	353,100	400,000	400,000
Midway	50,000	29,200	910	5,000
McKittrick	639,500	1,353,500	1,875,925	720,000
Newhall and Ventura	626,540	683,500	663,100	500,000
Fullerton and Brea Canyon	1,195,015	1,427,700	147,500	1,750,000
Whittier and Puente	687,030	878,015	748,000	960,000
Summerland	94,550	131,000	120,000	75,000
Sargents	....	....	35,090	20,000
Halfmoon Bay	....	....	1,000	2,000
Arroyo Grande	....	....	....	5,000
Total	13,973,500	24,337,828	28,476,025	35,671,000

It was thought that California oil, because of its asphaltum base, could not be refined for kerosene at a profit, but this fallacy has been disproved, for most of the kerosene used on the Pacific Coast today is made in or near San Francisco. California's petroleum production has risen until its output in barrels is greater than that of any other State, and while gold is still the leading product in point of value in the mineral industry of the State, petroleum now ranks second.

The production in 1905 was 35,671,000 bbl., against 28,476,025 bbl. in 1904, and only 4,000,000 bbl. in 1900. Of the output in 1905, over 800,000 bbl. of crude oil were exported, about 4,000,000 bbl. were manufactured into kerosene, and 10,000,000 bbl. remained in tankage. The railroads of the State are said to be consuming about 15,000,000 bbl., and much of the remainder goes for sprinkling roads and manufacturing lubricating oils, and naphtha.

Kansas and Indian Territory.—The history of developments in these fields in 1905 is related in the paper by Professor Haworth, which follows this review. The monthly pipe line runs, etc., are given in the following table:

MONTHLY PIPE LINE RUNS OF THE PRAIRIE OIL AND GAS COMPANY, 1905.
(In barrels.)

Month.	Runs.	Daily Average.	Deliveries.	Daily Average.	Put into Tanks.	Daily Average.	Total Stocks.
January	793,648	25,602	315,426	10,175	478,222	15,427	5,655,672
February	564,482	20,160	292,649	10,452	271,833	9,708	5,912,207
March	695,908	22,449	249.564	8,050	308,087	9,938	6,327,923
April	549,338	18,311	241,252	8,042	308,087	10,270	6,657,926
May	784,229	25,298	219,065	7,067	563,164	18,231	7,256,628
June	715,397	23,847	185,889	6,196	529,508	17,650	7,573,536
July	1,091,000	35,194	206,155	6,650	884,844	28,543	8,265,004
August	1,212,912	39,126	279,391	9,013	933,521	30,114	9,213,216
September	1,203,362	40,112	318,864	10,629	884,498	29,483	10,618,676
October	1,380,208	44,522	209,094	6,745	1,171,114	37,778	11,585,178
November	1,355,012	45,167	416,476	13,883	938,535	31,285	12,510,152
December	1,509,325	48,688	684,642	22,085	824,683	26,603	13,250,118
Total	11,854,821	32,373	3,618,467	9,916	8,096,096	22,086	104,826,236

Illinois.—The new Westfield oil district in Illinois has shown rapid and satisfactory development, although not on a large scale. The field extends through Coles, Cumberland, Clark, Crawford, and Jasper counties to Robinson and, from east to west, it extends from Toledo nearly to the Wabash river. Oil and gas are found at depths from 200 to 900 ft., the southern wells being deepest. Near Westfield, where the territory is well defined, every well put down has shown either gas or oil. The oil is said to be better than Kansas and as good as that of the Ohio-Indiana fields, selling for 81c. per bbl. The wells are all pumping wells and 21 of them in a recent month averaged 20 bbl. each per day. Wild-catting is now going on at a number of points within the State. Individual companies operating in the field have a production of 400 to 500 bbl. per day.

Louisiana.—The Louisiana field increased its production in 1905, due to the heavy flow in the Jennings field. In this district, 27 wells were drilled during the year, causing an increase in the production of over 3,000,000 bbl., beside making good the decrease in the output of the older wells, though the staying qualities of the Louisiana wells are superior to those of Texas. The Texas company expects to complete its pipe line to Lake Charles, a distance of 35 miles, early in 1906. This

line will be of great benefit, as it will enable the company to send shipments by other railways than the Southern Pacific.

Louisiana's total production for 1905 was 9,672,015 bbl. as compared with 6,611,419 bbl. in 1904, which shows an increase of nearly 50 per cent.

Ohio and Indiana.—In Ohio and Indiana, as in Pennsylvania, the conditions were not so favorable in 1905 as during the year preceding, and in the face of advancing prices the production materially decreased. All efforts to find new pools were unsuccessful, while efforts to extend the present developed territory were rewarded by comparatively little success, as is evidenced by the high percentage of failures to find oil, about every fourth well proving dry. The combined production of oil in the Ohio-Indiana field for 1905 was somewhat over 22,000,000 bbl., as compared with 24,000,000 bbl. in 1904, a decrease of 8 per cent. in the later year.

Pennsylvania.—This State is failing to produce just when her high-grade oil is needed most in the manufacture of gasolene and the lighter products of petroleum, for which there is a heavier demand than ever. The yield of these products cannot be augmented from the heavy asphalt oils produced abundantly in Texas and Louisiana.

Further details of the petroleum industry in Pennsylvania in 1905 will be found in the paper by Professor George, which follows this review.

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS IN THE APPALACHIAN FIELD.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Yearly Average.
1899	$1.17	$1.15	$1.13	$1.13	$1.13	$1.13¼	1.22½	$1.27½	$1.44½	$1.50⅞	$1.57⅜	$1.65⅛	$1.29⅜
1900	1.66⅔	1.68	1.68	1.55	1.39⅜	1.25½	1.25⅝	1.25½	1.23	1.10⅜	1.06¾	1.08¾	1.35¼
1901	1.69½	1.25	1.29	1.20½	1.07⅜	1.05	1.13⅜	1.25	1.25⅜	1.30	1.30	1.21	1.21
1902	1.15	1.15	1.15	1.17½	1.20	1.20	1.22	1.22	1.22	1.28	1.38	1.49	1.24
1903	1.52	1.50	1.50	1.51½	1.51½	1.50	1.53	1.56	1.56	1.69½	1.79½	1.88½	1.59
1904	1.85	1.82½	1.73	1.65	1.59½	1.57	1.53½	1.50	1.54½	1.56	1.58	1.55	1.62⅜
1905	1.43½	1.39	1.37½	1.32½	1.28	1.27	1.27	1.27	1.39	1.58½	1.59½	1.58	1.31⅜

Texas.—The Texas field increased its production during 1905, due to the discovery of the Humble field, where a gusher showing an initial production of fully 7000 bbl. per day was brought in, and the excitement following was only second to that after the discoveries at Spindle Top. The people, however, having gained experience, developed the new field on rational lines, the result being a rapid increase in production; but during the second half of the year the production decreased, despite the effort to extend the field.

The total production in 1905 was 30,354,263 bbl., the greatest yield of oil ever credited to the State in a single year. All but 595,306.82 bbl. of the Texas output came from the Gulf Coast districts in the southeastern

part of the State, which produced 29,808,956.86 bbl. against 21,520,175 bbl. in 1904, an increase of 8,288,781.86 bbl.

The close of the year, however, witnessed a decline in the State's production, and especially in the Humble field, where in December the output was less than 20,000 bbl. a day as against 100,000 bbl. a day in June. During the 12 months of 1905, the Humble district put out 18,066,428.22 bbl. of oil—the greatest quantity ever produced by a southeastern Texas district, not excepting Spindle Top, which produced 17,420,949 bbl. in 1902.

The banner month for production was June, when southeastern Texas made a record of 4,150,090.35 bbl., and prices for oil at the wells went as low as 13c.

Unless new fields are soon discovered in Texas, the production of petroleum will decrease very rapidly, for it is characteristic of the wells of this State to be short-lived. Even at the present time the storage is being drawn on to fill shipping demand, and as much of this stored crude is in earthen tanks, and has been out of the wells many months, it has lost a large percentage of the lighter elements, and holders are anxious to sell. It is estimated that producers in Texas and Louisiana have lost over a million dollars through the evaporation or seepage from earthen tanks. Millions of dollars have been squandered in Southern Texas, due to mismanagement and folly, and only a few well-managed, legitimate companies have made money. Oil has been forced out by compressed air and the wells exhausted in a few months, resulting in crude being sold for a fraction of what it was worth.

Petroleum Production in Foreign Countries.

Austria-Hungary.—The exports of petroleum products from Austria-Hungary to various countries in 1905, according to official statistics, were as follows (in metric tons):

Destination.	Illuminating Oil.	Gas Oil.	Lubricating Oil.	Benzine.	Total.	
					1905.	1904.
Germany......	82,044	526	14,581	2,000	99,151	79,332
France........	26,508	5	1,877	925	29,314	9,193
Turkey........	27,510	—	95	3	27,608	8,877
Switzerland....	8,125	220	2,758	3,781	14,885	13,935
England.......	10,312	—	2,704	—	13,016	6,779
Holland.......	509	108	7,227	—	7,844	4,382
Italy..........	1,702	65	2,588	1,109	5,464	4,810
Belgium.......	124	60	3,843	—	4,027	3,321
Bulgaria.......	2,627	1	23	3	2,654	921
Other countries	2,253	67	1,081	364	3,765	4,574
Total......	161,714	1,052	36,777	8,185	207,728	136,124

The total exports of the different products in 1904 were as follows: Illuminating oil, 91,500 tons; gas oil, 2391 tons; lubricating oils, 28,-

527 tons; benzine, 13,706 tons; total, 136,124 tons. The total exports of illuminating oil to Germany during 1905 were 33,197 tons to Hamburg, and 48,847 tons to the rest of Germany. In 1904, the respective figures were: 10,470 tons to Hamburg, and 46,856 tons for the rest of Germany. A large part of the petroleum shipped to Hamburg, however, was destined for Holland and Belgium, which countries in reality consumed more Austrian illuminating than is shown in the above table of exports.

In addition to the above, there were exported from Austria-Hungary in 1905, 2220 tons of unrefined paraffin scale, against 1152 tons in 1904; refined paraffin scale, 6770 tons in 1905 against 4840 tons in 1904; ozokerite, 1614 tons, against 2093 tons; and ceresin, 831 tons, against 676 tons.

Galicia.—The oilfields in Galicia showed a slight decline in 1905, falling from 827,116 tons in 1904 to 801,796 tons in 1905. The steady growth of the yield, which began in 1900, has for the moment come to a standstill. It would be wrong, however, to decide that the production of crude oil in Galicia has reached its highest limit, and that hereafter it will show a decline. It was only in 1901 that the production began to increase, with the development of the Boryslaw field. In the same manner as the development of the Schodnica field sent the output up in the nineties, so the opening up of Boryslaw shows its effect on the total output since the beginning of the new century. The output of the Boryslaw field has been as follows:

	Tons.		Tons.		Tons.		Tons.
1898	13,000	1900	55,000	1902	226,000	1904	546,000
1899	18,000	1901	132,000	1903	373,000	1905	546,500

In the last two years, the output of Boryslaw has included the production of the neighboring field of Tustanowice. The above figures show that the decline in the total production for the year was not caused by any decrease in the output of the Boryslaw-Tustanowice field, as the latter shows even a slight advance. The cause of the decline is to be looked for in the other fields, the output of which in the last two years has been as follows:

Field.	1904.	1905.	Field.	1904.	1905
Potok	22,864	22,479	Other West Galician fields	34,411	35,607
Rogi	47,531	24,234	Schodnica	72,627	60,201
Rowne	2,454	1,609	Urycz	27,420	20,346
Tarnawa-Wielopole	10,707	32,956	Mraznica	4,915	3,646
Krosno	48,228	43,559	Other East Galician fields	9,909	10,600

The Schodnica field has proved to be one of the most durable, for after 15 years' continuous exploitation it is still yielding without any great efforts, mainly by the pumping of old wells, and still occupies the second place among the oil producing fields of Galicia.

Canada.—The development of the Alberta oilfield during 1905 was very active and attended by good results. This district is situated in the extreme southwest corner of the province of Alberta. Alberta lies just north of Montana, and its western boundary is the summit of the Rocky Mountains, which form the eastern line of British Columbia. Petroleum in this district was first made known by large springs which existed for years, as much as 900 gal. of oil being skimmed off the surface of the pools in one season, and sold for illuminating and lubricating purposes.

In 1901, John Lineham, a wealthy cattleman and lumber manufacturer, investigated and then bought up the field, putting in the latest machinery, and began work in earnest. After many breakdowns, delays, etc., a 12-in. hole, down 1100 ft., was completed late in the fall of 1905, and struck a large basin of oil. Mr. Lineham formed a company called the Rocky Mountain Development Company, Ltd., to develop the field. The company is preparing to build a refinery, and a pipe line to Lethbridge, 60 miles, which will give connection with two transcontinental railroads.

The freight rate is so high from the East that oil cannot be retailed in western Canada for less than from 40 to 60c. a gallon, and is never wholesaled for less than $10 a barrel, so with that and the 1½c. a gallon bonus from the Dominion Government the prospects are favorable. This oil has a paraffin base, and is 76 per cent. illuminating, 23 per cent. gasoline, and does not contain even a trace of sulphur. Its specific gravity is 36 degrees.

The output of petroleum in Canada in 1905, computed from the amount paid for bounties, was 634,095 bbl., valued at $1.34 per barrel. The bounty paid by the Dominion Government went into effect June 8, 1904, at the rate of 1½c. per imperial gallon. Ontario continues to afford the main supply; the Petrolea district has declined steadily, but newly discovered pools in Essex and Lambton counties have offset the diminution of the older field.

Germany.—The importation of petroleum into Germany from Austria has been steadily increasing, and from March 1, 1906, it is expected that this increase will be more marked, as from that date the special low freight rate, which up to the present has only been granted to Russian petroleum, is to come into effect for the Austrian product. The United States supplies the majority of petroleum used in Germany, but the figures remain fairly even, while the imports from Russia and Rumania show a decrease, only those from Austria-Hungary showing any proportionate increase.

As compared with its imports the production of petroleum in Germany is insignificant; the output in 1904 was 86,620 metric tons, and in 1905 it was 78,869 tons.

Persia.—Consul Norton, of Smyrna, Turkey (*Daily Consular Report*, July 31, 1905), calls attention to the petroleum-bearing rocks in Mesopotamia, west of the Persian frontier. A company (having the franchise for the construction of the Bagdad railroad) has the concession for the exploitation of these oilfields. Engineers and oil experts have made flattering reports of the extent, nature and possibilities of the Mesopotamian oil region. The importance of such a source of fuel to the railroad (in a country entirely destitute of other fuel) is sufficiently apparent.

There appears to be a broad petroliferous belt extending in a northwest-southeast direction from Kurdistan down through Mesopotamia, ancient Assyria and Chaldea, and the adjoining portions of Persia to the Persian Gulf. There are indications that this belt is prolonged to the east, passing through Baluchistan into India. In the Persian section, the surface indications are excellent. Bitumen is of frequent occurrence. Crude petroleum often oozes from the ground; and on certain streams, layers of oil are encountered several inches thick. The natives often dip, from the surface of these streams, the oil which they require for use as a combustible or an illuminant.

Peru.(By V. F. Marsters.)—Of late years there has been an increasing interest in the development of the oilfields of northern Peru. For many years it has been known that lubricating and fuel oils existed at several points in the provinces of Paita and Tumbes. Some of these have been developed in a commercial way within the last 10 or 15 years. The oldest fields now placing oil on the Peruvian markets are those situated at Nigritos and Zorritos. At Lobitos, however, about 20 miles north of Nigritos, and on the coast, is a new field recently developed by the Peruvian Corporation. In this locality exploratory work has been carried on for nearly three years, on a somewhat modest scale. The first work in the field was attended with various difficulties, which seriously hampered and interfered with rapid exploration. At the present date 17 wells have been put down, and eight of them are now producing oil. The most productive ones have been bored during the last 10 months. The oil-bearing sand is tapped at a depth of about 1000 ft. Above the sands the entire series of deposits is made up of very fine clays. In such of the wells as have penetrated the oil sand, the fine clay again appears. Only a few feet has been penetrated below the oil horizon. From the structure of other fields it would seem quite probable that other oil sands may be found at a greater depth. No effort has been made, however, to demon-

strate this suggestion. There are 1000 tons of tankage, all of which is filled with oil. The product already on hand is soon to be placed upon the Peruvian markets. Sufficient work has been done and results obtained to place this field on the list of producers in Peru.

Rumania.—The petroleum industry of Rumania progressed steadily during 1905, and the advance was considerable. The increase in the production in 1905 was considerably over 20 per cent.

Russia.—The production of petroleum in Russia in 1905 was approximately 400,000,000 poods, or, more nearly, 6,500,000 tons. This decrease of more than 4,000,000 tons as compared with 1904 is easily understood when the conditions which clouded the Russian industry are recollected. It is very gratifying that the losses, due to the Russian disturbances and the wholesale destruction of oil at Baku, were not much greater. Early advices and reports from Baku were very alarming, and it was thought improbable that any other producing field could counteract the effect of these unfortunate disorders.

The work of restoring the damaged properties in Baku is well under way, and the production is again steadily increasing. In order to facilitate the rebuilding, the Government is making loans to the producers and refiners who lost heavily during the disasters, and the restoration of the field to its former productiveness will be more rapid than was originally anticipated. According to good authority the destruction of the majority of the oil properties in the Baku district was not without good effect, which will probably show in the direction of technical improvements, of which many of the properties were badly in need. The erection of new oilfield plants is being carried out on certain improved lines, particularly in the application of internal combustion engines and electric power. After the completion of the work of reconstruction, and all the properties are again working at their full capacity, it is expected the output will not only return to its normal rate, but will greatly exceed it.

The naphtha producers and manufacturers at Baku not agreeing with the report of losses, as estimated by the commission appointed by the Minister of Finance, appointed a committee of their own to study the question. The two reports may be summarized as follows:

	Government Report.	Naphtha Industries' Report.
Percentage of derricks destroyed	60	$58\frac{1}{2}$
Average output of destroyed derricks	27,200,000 poods per month	920,920 poods per diem
Money loss incurred by destruction of plant and naphtha	19,776,000 rubles	28,700,000 rubles
Appreciation of material required to reinstate plant	....	13,300,000 rubles

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called) to Coody's Bluff, 10 miles north of Alluwe, forming a north-and-south trend known to be from 15 to 18 miles long. Here oil of good quality is found at depths varying from 300 to 650 feet. In daily capacity, the wells range from 10 to 500 bbl.; most of them flow from one to six weeks without pumping. This is the most important shallow-sand territory yet developed in the entire mid-continental area.

At Cleveland, the oil-sand is very deep (around 1900 ft.), with slight variations in different wells. The production ranges from 25 to 300 bbl. per well initial capacity, making this an important oil pool.

Another new pool opened entirely in 1905 is located to the north and west of Dewey, four miles north of Bartlesville. The first development here was made by Stubbs and Low, early in the year, on a lease lying 1½ miles northwest of town. The oil-sand averages 1400 ft. below the surface; although a shallower sand, about 800 ft. down, is a small producer in some of the wells. From this original development, in almost all directions, new wells are frequently brought in. Early in November, a well two or three miles southeast of Dewey started flowing at the rate of 1000 bbl. per day; at the end of a week it had settled to 450 bbl. A few good producers have been brought in still further northeast toward the Kansas line; but up to the close of the year there was left an unexplored zone nearly 10 miles wide between the developed area and the southern Kansas line, with good reasons for believing it may become as productive as any area yet developed.

The most extensive development throughout the year was in the areas fairly well proven during 1904. From the vicinity of Bartlesville southward, along the extreme eastern line of the Osage territory, a surprisingly large number of strong wells were brought in. A number of them have begun flowing at the rate of 1000 bbl. per day, and have held weeks before dropping to 500 bbl. For matters of convenience, holders of the blanket lease on the Osage lands divided the eastern area into three north-and-south columns, approximately three miles wide per column. East-and-west lines were now drawn half a mile apart; each sub-area is called a "lot," and is numbered from the State line southward to the south side of the territory, a distance of 58 miles, making 116 lots in each tier. Practically all the land extending from Bartlesville to lot 70 or 75 is proved territory, with richer areas in certain places. Lots 44 and 45 are excellent, with another rich pool about lots 51 and 52; the richest of all is in the vicinity of lots 61 to 67. Here it seems almost impossible to find a dry well; numerous wells had initial capacities above 300 bbl. per day. West from this immediate area, on the middle tier of lots (that is, lots 169 to 172), there is also a productive area; and still west on the third tier (that is, lots 293-296), and in parts of Township 24, Range 10, still farther west, excellent oil pools have been developed. It seems probable

that here is an east-and-west trend reaching from the east side of the Osage leases westward, a distance of 15 miles. The north-and-south development from Bartlesville to lot 68 is another stretch of excellent land, 18 miles long. East of the territory line, and within the Cherokee land in the vicinity of Ramona, a number of dry wells have been drilled, implying a limitation of the field on the east side.

Outside of the richest areas already described, other excellent pools have been developed in part. Thus, a zone reaches west and a little north from Bartlesville over into Range 10, which is dotted here and there all over with good wells. To the southwest across the river north from Cleveland a number of good wells are found; again, in the vicinity of Pawhuska, both oil wells and gas wells are found, which implies a profitable area. The development in different parts of the Indian Territory has been greater than to the north in the State of Kansas; due in part to better wells, and in part to legal considerations to be explained later.

In Kansas, development was not great during 1905. The Chanute-Humboldt pool, the Bolton pool, the Coffeyville pool, the Tyro pool, and the Caney pool were tolerably well defined before the close of 1904; then a severe winter came on, and just at the beginning of milder weather the price of oil dropped so low that producers had no incentive to drill. In fact, for about six months (from February to July inclusive) it was impossible to market more than a small fraction of oil which might have been produced from the wells drilled. As a result, surprisingly few wells were drilled in Kansas. Chautauqua county witnessed more drilling than any other part of the State outside of Paola and Rantoul, already mentioned; partly because the grade of oil was highest here and commanded the highest market price; and further, on account of less uncertainty in drilling. Three miles straight south from Sedan, the county seat, is a pool known as the Huffman pool. Oil is obtained there at from 1050 to 1250 ft.; the wells range from 20 to 256 bbl. initial capacity, many of them making 75 bbl. after months of pumping. Two miles to the east, and from two to six miles south of the Huffman field, is an area trending north and south, about two miles wide and reaching down to the State line, commonly known as the Blundell-Spurlock area. This was the first developed in Chautauqua county outside the immediate environs of Peru; on the whole it has proved to be a rich field. Immediately south of Peru about three miles another pool developed during 1905, known as the Scott pool. There are nearly forty wells along a north-and-south line about two miles long, with dry wells on the east and light gas wells on the west. There seems to be a north-and-south trend of gas territory between this and the Blundell-Spurlock pool.

Production.—The production for the year 1905 was about 12,000,000 bbl. The table given on a previous page shows the total production

by months of the entire area, as bought by the Prairie Oil and Gas Company and according to its monthly report: it exhibits the total runs daily; average of the same; total deliveries to refineries; daily average of same; total put into tanks; daily average of same; and, finally, total stocks on hand at the end of the year and for each month during the year.

To these should be added a small amount consumed by the independent refineries at Humboldt, Paola, Cherryvale, etc., and the still smaller amount used as fuel and lubricating oils, which may be estimated at from 100,000 to 125,000 bbl.

It will be noticed that, beginning with July, the monthly runs exceeded 1,000,000 bbl.; and that for the last three months it exceeded one and one-third million; so that 1905 turns over the industry to the new year on a basis of more than 16,000,000 bbl. annual production. Of this amount, much more than one-half came from the Indian Territory; but at the present an exact division cannot be made. Presumably, however, not more than one-third of it came from Kansas.

The extra development in the Indian Territory, as already explained, was due in part to stronger wells and less uncertainty in drilling. But another very important factor is the legal status of leases on Indian lands. The Secretary of the Interior requires a certain form of lease, one clause in which specifies that at least one well shall be drilled by the end of the first twelve months after the lease is approved. This has been the most important factor in forcing drilling. The Secretary is of the opinion that he is best serving the cause of the Indian by giving him royalties from oil wells as rapidly as possible. It is generally believed that the low price of oil is due to the great excess of development over consumption. According to the table given, the Prairie Oil Company has 14,000,000 bbl. of oil stocks stored, here and there, throughout the oil-producing territory and at the Sugar Creek refinery near Kansas City, with an increase of nearly 40,000 bbl. per day. With the entire area controlled by one good business head, development would be stopped until the stocks had decreased greatly and the price of oil advanced; but with the Secretary of the Interior insisting on the "development clause," no one cares to jeopardize the title of valuable leases by declining to drill; therefore the price goes downward, and production continues to increase.

Gas.—The production of gas was greatly increased during 1905. Early in the year the Kansas Natural Gas Company began buying property, and has continued that policy to the present. It now owns all the leases formerly owned by almost every big gas company in the State, and stands ready at all times to pay for any new developments that may be made. It now owns nearly all the good production of Montgomery and Wilson counties, the two richest gas fields in the mid-continental area. From near Independence south to the State line, wells with a daily capacity

of 15,000,000 cu. ft. are comparatively common, and some producing 30,000,000 are known. One well measured when open (by using the Pitot tube, the mouth of which was placed a little to one side of the center of an 8¼-in. pipe) gave a pressure of 38 lb. on the gage attached; this is perhaps the largest well in the entire region.

A well drilled during October near the State line tested close to 35,000,000 cu. ft. In the Indian Territory some enormous gas wells likewise have been found, a number of which range from 15,000,000 to 20,000,000 cu. ft. The strongest wells are near the line between the Osage and Cherokee lands, some on one side and some on the other. In the vicinity of Bartlesville, gas is found in the sand above the principal oil-sand. Frequently this gas is allowed to escape and the drill sent on down to the oil. In other places gas is in a sandstone corresponding to the oil-sand. Recently a number of strong gas wells have been drilled two or three miles southeast of Bartlesville. Some of the strongest wells are found west of Ramona; in the vicinity of Pawhuska wells ranging from 8,000,000 to 15,000,000 cu. ft. have been obtained.

The Kansas Natural Gas Company now has its pipe lines laid northward beyond the Kansas river almost to Atchison and St. Joseph, with one branch leading west to Topeka and another line to Kansas City. A large pumping station is established at the little town of Petrolia, on the west side of the Neosho river, about midway between Humboldt and Chanute. Pipes from all over the gas field to the south converge at this point. Two 16-in. pipes are laid, side by side, from the pumping station to Ottawa; here they diverge, one going to Kansas City; the other, north to Lawrence, Leavenworth and Atchison. About three miles south of Lawrence a side line is carried westward to Topeka.

Gas is sold throughout these northern towns at 25c. per thousand for domestic consumption, measured at about 8 oz. above atmospheric pressure. For manufacturing purposes it is offered at lower figures; but up to the present few factories have adopted its use, as gas cannot compete with coal for manufacturing purposes, unless sold considerably below any figures yet offered. All the important gas towns in Kansas and Indian Territory are still offering 3c. gas to manufacturers; some of them in the Indian Territory are offering it at a lower price.

Refineries and Pipe Lines.—The Standard Oil Company has a small refinery at Neodesha, with a capacity of about 3500 bbl. crude per day, which was kept in operation throughout the year at its full capacity. It has built a larger refinery at Sugar Creek, in the eastern suburbs of Kansas City, with a reported capacity of 6000 bbl. per day, and which began receiving oil early in the year. One 8-in. pipe-line leads to the Sugar Creek refinery from Neodesha, with many laterals ramifying through the entire oilfield to the south. At present, a second 8-in. pipe-line is

building to Sugar Creek. Also, an 8-in. line reaches from Sugar Creek to Whiting, Indiana, which was open for operation during the latter part of the year.

Large tank-farms have been established at different places throughout the field and at each refinery, the largest one at present being at Neodesha, with smaller ones at Humboldt, Caney, Bartlesville, Cleveland, Ramona and perhaps other places. Tanks are still building at a rapid rate in order to keep ahead of production.

A number of small independent refineries are located in the State, one at each of the following places: Humboldt, Paola, Cherryvale, Niotaze and Longton. Those at Humboldt and Paola have been in operation the greater part of the year consuming an aggregate of about 300 bbl. per day. The others have not yet done much refining and are small.

Markets.—Until early in the year 1905 the Prairie Oil and Gas Company made a somewhat arbitrary division of the Kansas oilfield for the purpose of establishing prices. In all market quotations previous to this time, Kansas oil was graded as South Neodesha and North Neodesha, with the former bringing regularly 20c. a bbl. more than the latter. The highest price paid in the field was $1.38 for South Neodesha oil early in the year 1904. This quotation lasted but a few days, when the price began to drop and gradually declined throughout the year, reaching 72c. in December. The downward tendency continued until it reached the low price of 50c. a barrel for the best Kansas-Indian Territory oil during the first half of 1905, at which point the market rested for months. In the middle autumn, a 2c. advance was made, after which the market was stationary to the end of the year.

Early in the year the old-fashioned way of grading oil into South Neodesha and North Neodesha was abandoned, and a gravity scale was adopted. Under this regulation, oil testing 32 deg. B., or more, was given the top price, and a decrease of 10c. per bbl. for each degree, or 5c. for each half-degree, was put upon the lower grades of oil. Gravity tests were made by the gager and were strictly enforced, often a shade of one-tenth degree causing a difference of 5c. per barrel.

A portion of the oil from Montgomery county, all the oil from Chautauqua county in Kansas, and practically all from the Indian Territory, Oklahoma field, tests 32 deg. B. or more, some of it reaching as high as 37 deg. B., or 38 deg. B. Almost all the remaining Kansas oil tests below 32 deg. B. Although a penalty of 5c. for each half-degree was imposed on oil below 32 deg. B., yet a corresponding increase was not made for that above 32 deg. B., so that wells producing oil testing from 35 deg. B. to 38 deg. B. have never been more profitable than they would have been had their product been barely above 32 deg. B.

Future Developments.—The success attending present operations in

the Indian Territory is so great that future development will probably increase over the present production. Should the price of oil start upward with a fair promise of reaching $1 per barrel, almost every lease holder in the oil region would begin drilling. Under such circumstances it is difficult to tell what the result would be. Could we have another year of such prices with a demand for the entire production, it would not be surprising if the field should reach a capacity anywhere from 75,000 to 100,000 bbl. per day.

Pennsylvania Oilfields.

By Harold C. George.

The production of petroleum in western Pennsylvania reached its maximum in 1891, when the prolific McDonald field turned out such remarkable gushers. Since then, with but few exceptions, there has been a continuous decline in the production of all fields of the State; and it is only the continual drilling of new wells and the cleaning out of old ones that prevent this decline from being more rapid.

The oilfields of the State are naturally divided into three groups, as follows:

First, the northern district, which includes the Bradford field (of McKean county), the Warren, Clarendon and Cherry Grove fields of Warren county, and the fields of Elk and Forest counties.

Second, the central district, which includes the Venango, Clarion, Armstrong and Butler county fields.

Third, the southwestern district, which includes the fields of Allegheny, Washington and Green counties.

In the fields of Bradford and Warren counties little drilling is being done, and the production is gradually decreasing. At the present time the average production of these fields is about ¼ bbl. per day for each well. Most of the wells have been drilled from 15 to 30 years; and, as the sand is compact and uniform, the wells have enjoyed a long life.

In the fields of Elk and Forest counties drilling for oil has not been as extensive as formerly, but operations for natural gas are being carried on with good results. In these fields the production is better than in the fields of Bradford and Warren counties, as the wells are not so old and they are worked more systematically.

The field of Venango and Clarion counties is in much the same condition as the northern fields. Little drilling has been done this year, and the production is declining gradually.

In the Butler county fields drilling has been carried on more extensively; and, although the "territory" has been drilled over many times, there are still some pools left to be found by some fortunate prospectors. The "territory" is far from being uniform, and the wells are not long lived.

Many of the oldest men in the business have been deceived by the wells of this region. The recently drilled "Spotty McBride" well might be mentioned in this connection as the most deceptive of them all. This well was drilled during the summer, near Butler, and at first was a gusher; but today, of the 30 wells drilled in this pool since the striking of the original gusher, less than one-third are producing, and with an aggregate of only about 50 bbl. per day; of this amount the original well is said to produce 15 bbl.

The Armstrong county field shows a revival of operations, but the southwestern fields have the most marked gains.

Some good wells have recently been drilled in Allegheny county, and present operations promise good results.

The operators in the older fields have generally applied themselves to the more economical production of petroleum from the old wells, rather than to the drilling of new ones.

The cheap source of power offered by the natural gas engine has aided much in operating, and in prolonging the life of, the wells. It is almost universally used in all the fields of the State, and by its great economy, and its capacity to run without a constant attendant, it has become profitable to pump small wells where a number of them can be operated from a centrally located "power." Under other conditions, such wells would have to be abandoned.

The great trouble with so many of the old wells is due to the wearing out of their tubing, lead-lines and rods. It does not pay to purchase a new equipment for such wells, and hence they are either abandoned, or else a boring rig or some other cheap method of operating is installed.

On the 1st of January, 1905, the market price of Pennsylvania crude petroleum was $1.50 per bbl.; but it did not remain long at that point and continued to decline until it reached $1.27, where it remained until the middle of September, when it began to advance rapidly until 34c. had been added. This advance has had a marked effect on the industry. There is renewed activity in all the older sections of the Pennsylvania fields. There seems to be no territory too small to invite the attention of the producer, providing there is a reasonable expectation of finding oil. Wells that a few years ago would have been abandoned without any attempt to convert them into valuable producers are now in urgent demand; and the output, however small, is carefully looked after. It is evident, however, that no material increase in the supply of high-grade petroleum can be expected from these sources. There is no doubt that the recent increase in the market price of oil has stimulated field developments; but it is purely problematical whether this stimulus will be sufficient to increase the production so as to meet the demand.

The old fields have been very closely drilled, and the old wells have prac-

tically drained adjacent territory which itself might formerly have been advantageously drilled. The most noticeable effect of the advance in the market is seen in the old fields and in shallow territory. Wells which were neglected during the depression are being put into better shape so as to increase their yield. Yet all the renewed production of the old wells, and all the drilling in the State during 1905, have failed to sustain the production; and the prospect for discovering new producing pools is far from encouraging.

The gradual decline of the Pennsylvania fields, and the more rapid failure of the gusher territory of West Virginia (which for years has been the principal factor in the yield of high-grade oil), have decreased the supply just when the demand is greatest.

The increasing use of gasoline and of the lighter products of petroleum has brought about a heavier demand upon the fields producing high-grade oils. The manufacture of these products is naturally limited, and it is impossible to augment the yield from the heavy asphalt oils produced abundantly in Texas and Louisiana.

Although the "wild catter" has been industriously seeking after new pools and extending the limits of the old ones, the large list of "dry holes" for the year bears witness to the fact that there is no longer anything of value in reserve for the drill. In the older fields the activity is confined to locations that would be passed by if there were anything better in view.

An oil well begins to die the day it is born; so that it is only a question of days, of months, or of years until the end comes. This might be said of the Pennsylvania oilfields in general. Their decline has been going on for several years, and one of two things is destined to happen in the near future: Either there will be a marked advance in the price of high-grade petroleum, or else the production of the State will rapidly decline.

PHOSPHATE ROCK.

THE shipments of phosphate rock in 1905 amounted to 1,933,286 long tons, valued at $9,713,296, as compared with 1,874,428 tons ($6,873,625) marketed in 1904. Foreign demand, as in the earlier year, consumed about two-thirds of the production. Each of the three regularly producing States showed an enlarged output in 1905, the increase being more striking in Florida and Tennessee than in South Carolina.

Market and Prices.—The market for phosphates was active throughout the year and prices showed a general advance. Florida high grade rock, analyzing 77 to 80 per cent. bone phosphate of lime, rose to $7.25@7.50 per long ton, f. o. b. Fernandina, Fla., and $10.88@12.25 per ton, c. i. f. European ports. Land pebble, 60 to 73 per cent., likewise advanced to $4@4.25 per ton, f. o. b. Fernandina, and $7.90@8.60 c. i. f. European ports. Tennessee rock for export, 78 to 80 per cent., was quoted at $4.60@4.75 per ton, f. o. b. Mt. Pleasant, and $10.79@11.19 c. i. f. European ports. Domestic 78 per cent. rock, $4@4.25, f. o. b. Mt. Pleasant, and lower grades, $3.25@3.75. At the close of 1905 South Carolina land rock, 58 to 63 per cent., was quoted at $4@4.25 per ton, f. o. b. vessels, Ashley river; while river rock, 54 to 57 per cent., brought $3.75@4. A small export trade has been done with France and Great Britain at about $6.50@7 c. i. f.

IMPORTS OF FERTILIZERS INTO THE UNITED STATES.
(In tons of 2240 lb.)

Year.	1902		1903		1904		1905	
	Quan.	Value.	Quan.	Value.	Quan.	Value.	Quan.	Value.
Guano	8,407	$164,783	21,007	$251,966	35,876	$478,388	25,651	$365,823
Crude phosphates	137,386	646,264	132,965	697,112	130,214	745,744	56,021	273,289
All other fertilizers	....	1,725,333	....	2,353,496	....	2,856,141	....	4,051,003
Of which— Kieserite and Kainite	225,413	1,016,032	158,313	773,758	218,957	1,050,082	....	....

In the following table, phosphatic materials alone are included:

STATISTICS OF PHOSPHATES IN THE UNITED STATES. (*a*)
(In tons of 2240 lb.)

Year.	Production	Imports.	Exports. (*b*)	Consumption.	Year.	Production	Imports.	Exports.	Consumption.
1896	937,372	21,746	496,168	462,950	1901	1,483,723	180,714	729,539	934,898
1897	1,007,367	12,950	551,046	469,271	1902	1,600,813	145,793	802,086	944,520
1898	1,257,645	71,388	570,948	758,085	1903	1,581,576	153,972	785,259	950,289
1899	1,663,476	118,613	867,790	914,299	1904	1,874,428	166,090	842,484	1,198,034
1900	1,527,711	144,006	619,995	1,051,722	1905	*c*1,933,286	81,672	934,940	1,080,018

(*a*) Production statistics of 1901 and subsequent years, except 1905, are those of the Geological Survey and are based on marketed products. (*b*) Neglecting the insignificant re-exports of foreign product. (*c*) Shipments.

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES. (*a*)
(In tons of 2240 lb.)

Phosphate.	1902.		1903.		1904.		1905. (*b*)	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Florida hard rock..	429,384	$1,743,694	214,876	$1,988,243	531,087	$2,672,184	579,228	$4,198,210
Florida land pebble	350,991	810,792	390,882	885,425	460,834	1,102,993	401,997	1,660,248
Florida river pebble	5,055	9,711	56,578	113,156	81,030	199,127	90,065	371,968
Total Florida....	785,430	$2,564,197	860,336	$2,986,824	1,072,951	$3,974,304	1,071,290	$6,230,426
S. Car. land rock...	245,243	$753,220	233,540	$721,303	258,806	$830,117	221,712	$915,671
S. Car. river rock...	68,122	166,505	25,000	62,500	12,000	31,200	30,284	117,199
Total S. Carolina ..	313,365	$919,725	258,540	$783,803	270,806	$861,317	251,996	$1,032,870
Tennessee.........	390,799	$1,206,647	460,530	$1,543,567	530,571	$2,037,804	610,000	$2,450,000
Other States........	720	2,875	2,170	5,100	100	200	......	....
Total United States	1,490,314	$4,693,444	1,581,576	$5,319,294	1,874,428	$6,873,625	1,933,286	$9,713,296

(*a*) Statistics for 1902, 1903 and 1904 are those of the U. S. Geological Survey. (*b*) Statistics of 1905 are our own, and are based upon shipments, the values being based upon quotations at ports of shipment.

PRODUCTION OF PHOSPHATE ROCK IN THE WORLD.
(In metric tons.)

	1899	1900	1901	1902	1903	1904
Algeria..........................	324,983	319,422	265,000	305,174	320,843	343,317
Aruba (Dutch W. Ind.)............	(*c*)	(*c*)	(*c*)	10,530	15,511	22,764
Belgium..........................	(*a*) 190,090	(*a*) 215,670	222,520	(*b*) 135,850	(*b*) 184,120	(*b*) 202,480
Canada...........................	2,722	1,284	937	776	1,205	832
Christmas Island.................	(*c*)	(*c*)	(*c*)	61,178	70,096	71,757
France...........................	645,868	587,919	535,676	543,900	475,783	423,521
French Guiana....................	(*c*)	(*c*)	(*c*)	4,230	7,769	(*c*)
Norway...........................	1,500	300	738	2,295	1,795	1,456
Redonda (Br. W. Indies)..........	1,507	2,230	*Nil.*	132	1,102	1,729
Russia...........................	16,863	25,663	21,276	13,709	(*c*)	(*c*)
Spain............................	3,150	4,170	4,220	1,150	1,124	3,505
Sweden...........................	(*c*)	(*c*)	(*c*)	3,895	3,219	2,929
Tunis............................	......	....	....	264,930	352,088	455,197
United Kingdom...................	1,469	630	71	87	71	59
United States....................	1,539,953	1,515,075	1,507,548	1,514,159	1,606,881	1,904,419

(*a*) Cubic meters. (*b*) Metric tons of phosphate of lime; in addition there were 315,200 cu. meters of phosphatic chalk in 1902, 350,250 cu. meters in 1903, and 311,640 cu. meters in 1904. (*c*) Statistics not available.

Exports and Imports.—The total amount of crude phosphate exported in 1905 was 934,940 long tons ($7,465,592) as against 842,484 tons ($6,521,555) in 1904. Exports go, in order of importance, to Germany, Great Britain, Italy and France. The hard rock of Florida constitutes about half of the entire export. A large amount of Tennessee product is exported, but most of the South Carolina phosphate is consumed in this country. The ocean freight to European ports ranges from one-third to one-half of the cost of phosphate delivered at the American seaboard.

Imports consist principally of Peruvian guano, crude phosphate from the West Indies, and prepared superphosphates from Belgium. Imports of phosphates in 1905 experienced a sharp decline, as shown by the first table on page 488, in which the potash fertilizers are included merely for comparison.

Few new discoveries of merchantable phosphate rock have been reported

and, judging by the conservative prices paid for mineral land in Tennessee and Florida, the day of the speculator is closing.

Phosphate Production by States.

California.—The phosphate properties discovered in Wyoming and Idaho have been incorporated under the name of the Union Phosphate Company, in association with the Stauffer Chemical Company, of San Francisco. The former company will engage in the mining and shipping of crude rock, and in the manufacture of superphosphates on the Pacific Coast.

Florida.—In Florida the centralization of control of the larger mines by people who are mostly interested in foreign trade is becoming more and more important by reason of the higher cost of production and the smaller profit as a result of competition in European markets. During the winter and spring of 1905 the rainfall was very slight and the water level was lower than it had been since 1899. Owing to these conditions, the production of high-grade phosphate rock was unprecedented. Reverse conditions during summer and autumn made mining difficult and the production of the second half of the year was consequently below the average. The demand for high-grade rock at advanced prices increased, and the sales of the year were consequently augmented. Probably not less than half of the increase was made from stock carried over from 1904, with the result that at the end of 1905 the stock on hand in Florida was reduced to the lowest point in six years.

The total shipments by water from Florida in 1905 amounted to 1,071,290 tons, of which hard rock constituted 579,228; land pebble, 401,997; and river pebble, 90,065 tons. Of the total, 765,782 tons were shipped abroad.

Prices during 1905 were good, and on account of the rapidly increasing European demand, they promise to be even better in 1906. Buyers are already making contracts for 1907, and some are venturing to look as far ahead as 1908. The scarcity and increased cost of labor, and the resulting greater cost of mining, check the response that otherwise might be made to the greater demand and higher prices. The prices of high-grade phosphate rock during 1905 averaged from 25 to 50c. per ton more than in 1904.

South Carolina.—There is renewed activity in dredging for phosphate rock in the vicinity of Beaufort. The fact that the Central Phosphate Company is meeting with success in its operations on Coosaw river accentuates the opinion that there may be a revival in mining generally in the near future. One of the reasons adduced for this conclusion is that the Florida rock is nearing exhaustion. It is admitted that the Tennessee rock limit is in sight, while the phosphate rock in South Carolina's navigable streams is believed to be still very large. The present price of rock

furnished by the Central Chemical Company on Dale's creek, Lady's island, which is now producing from 800 to 1200 tons of rock out of Coosaw per week, is $2.25 per ton. It costs $1.50 per ton to mine it and put it on the road for foreign shipment. The royalty is 25c. per ton, and this leaves a profit of 50c. per ton. At one time, river mining in South Carolina was a prosperous industry, but in recent years production has fallen off heavily. Profits also depreciated, as a result of competition with phosphates produced in other sections of the country.

The total shipments from South Carolina in 1905 included 221,712 tons of land rock and 30,284 tons of river rock; none of the output was exported.

Tennessee. (By R. D. O. Johnson.)—The phosphates of Tennessee so far known to exist in commercial quantities are confined to Hickman, Maury, Lewis and Perry counties, in the center of the State. The brown-rock phosphate is a loosely coherent, porous, rusty-brown rock, disposed in rather thin plates resting directly upon one another or separated by thin layers of clay. It varies from 68 to 78 per cent. phosphate of lime, while its percentage of iron and alumina is generally over 5 per cent. The blue-rock deposit is confined to Hickman and Perry counties. In the latter county it is of such low grade that, under prevailing market conditions, it is not of commercial importance. This region has, however, been secured by the larger fertilizer companies. With the exception of small isolated areas on Leatherwood creek, Tuckers Bend on Duck river, and another place on Indian creek three miles south of Centerville, the workable beds of the deposit in Hickman county lie along Swan creek and its tributaries, and comprise the Swan Creek district. This district, with neighboring patches, has an area of 12,000 to 13,000 acres, about one-third being owned by the New York & St. Louis Mining and Manufacturing Company. The remaining two-thirds is held by the Virginia-Carolina Chemical Company, the American Agricultural Chemical Company, the Jarecki Chemical Company, the Meridian Fertilizer Company and the Tennessee Chemical Company.

The blue-rock phosphates are unchanged from their original form, and are, therefore, much harder and more compact than the brown-rock phosphates. They vary greatly in color and texture. The finer-grained, light-gray, compact varieties run much higher in phosphate of lime than the conglomerated, coarse-grained, dark-colored varieties. The grading of the rock is made in accordance with these characteristics. The rock varies from 55 to 75 per cent. phosphate of lime, and is sold under a guarantee that it contains less than 3 per cent. of iron and alumina. The thickness of the bed varies, from nothing on the edges to 50 in. at some places. Thickening of the bed is usually accompanied by coarser texture, and lower content of phosphate of lime. Under prevailing prices, a bed under 18

in. thick cannot be worked at a profit. The rock weighs 175 lb. per cu. ft. in the bed, or equivalent to 3400 long tons per acre per foot in thickness.

The deposits of brown rock vary greatly in thickness and lateral extent, but are cheaply and easily mined. Lying, as they do, under comparatively level fields or low hill tops, only a small amount of stripping is necessary to expose them. The rock is mined with the pick and shovel, no blasting being required. The nature, the high grade and the limited extent of the brown-rock deposits, together with their disposition and ease of mining, have contributed to heavy production, low prices and rapid exhaustion.

The impending exhaustion of the brown-rock deposits has been fully appreciated by large fertilizer manufacturers. While enjoying bountiful supplies of good phosphate at low prices, they have secured their future supply by getting into their possession the much larger, more certain, though somewhat lower-grade deposits of blue rock.

The region of greatest importance in the production of brown rock is Mt. Pleasant. The bed varies from 4 to 7 ft. thick and yields 1000 tons of merchantable rock per acre per foot of thickness. The upper third of the bed is usually dirty, loose and earthy, and is classed as "low-grade domestic," 68 to 70 per cent. phosphate of lime, with no guarantee as to iron and alumina. The middle third is harder and more compact, and is classed as "high-grade domestic." It analyzes 72 to 75 per cent. phosphate of lime, with 5 per cent. of iron and alumina as a maximum. The bottom third is much more solid, and is graded as "export"; it assays 75 to 78 per cent. phosphate of lime and from 3½ to 5 per cent. iron and alumina. It is a trade rule that an excess of 1 per cent. of iron and alumina over the guaranteed content is equivalent to lowering the grade of the rock 2 per cent. Blue rock, though generally lower in its content of phosphate of lime, holds a commercial parity by reason of its lower percentage of iron and alumina.

Mining can be carried on only in dry weather, as little or no provision is made for draining the cuts. The rock, as mined, contains from 15 to 20 per cent. moisture, and requires to be dried before shipment. The more primitive method is to spread the rock out over an even surface, depending upon the sun and wind to remove the moisture. The rock is turned periodically by plowing, and a single rain will undo the effects of a month's air-drying.

Kiln-drying is the method more generally employed. Cord wood is piled to a depth of 3 ft. over the area of the kiln, and upon this is placed the forked rock to a depth of 5 or 6 ft. It is confined, on the sides and ends, by dry walls laid up with the larger pieces of the rock. The kilns are generally under sheds, though many have no shelter. Some of the sheds are built with a kind of upper floor over which the loaded wagons

may be driven. Trap doors in the floor permit the filling of the kiln with but little labor. The wagons employed are fitted with gravel boxes having loose slat bottoms through which the rock is dumped to the trap doors. Drying costs 15c. per ton for cord wood, and 10c.per ton for building up the kilns.

The following is a statement of costs of mining and milling blue rock, based upon a bed 24 in. thick:

Mining, \$1.35 per long ton; hauling, \$0.35; crushing, \$0.25; loading in cars, \$0.025; total for crushed rock, \$1.975.

Ground rock—delivery to mill, \$1.53 per short ton; milling, \$0.50; loading, \$0.025; sacking, \$0.05; total, \$2.105. Sacks are extra.

Much of the ground rock is sold to the farmers and is used directly on the soil without further treatment. The "coarse crushed" rock is bought by fertilizer manufacturers.

The shipments of phosphate from Tennessee in 1905 have been estimated at 610,000 tons, of which only 80,298 tons were exported.

Progress in the Technology of Phosphates.

Electrolytic Acid-Phosphate.—An interesting regenerative process for the electrolytic production of bi-calcic phosphate as a fertilizer, from otherwise unserviceable raw-phosphate, is given by William Palmaer (of the Technical College, Stockholm, Sweden) in the report of the Canadian Superintendent of Mines (Part II, Annual Report, 1904). In substance, a solution of sodium chlorate (or per-chlorate) is electrolyzed to an anode acid-solution, and a cathode alkaline-solution. The raw phosphate is dissolved by the solution of chloric (or per-chloric) acid. From this, a bi-calcic phosphate (secondary phosphate) is precipitated, most of the sodium chlorate (or per-chlorate) being regenerated to be used again. It is calculated that not over \$1.50 per ton of finished phosphate is lost if the per-chlorate is used. It is claimed that one electric horse-power year produces 1.73 tons of bi-calcic phosphate with 36 per cent. soluble phosphoric acid; or 1.95 tons of the bi-calcic phosphate with 32 per cent. soluble phosphoric acid. The merits advanced are: First, the use of cheap low-grade material which is not suitable for the superphosphate industry. Second, the production of a high-grade (34 per cent. soluble acid) product. Third, saving of half the freight, from the high grade of the product. Fourth, little reversion of soluble to insoluble phosphoric acid occurs on storage. Fifth, the raw phosphate need not be finely powdered for treatment. Sixth, the bi-calcic phosphate can be used on sandy or boggy land when superphosphate is out of the question.

Manufacture of Fertilizer.—At Magdeburg, Germany, crude phosphate

rock is converted into fertilizer in the following manner: The rock is broken to coarse size and charged into a reverberatory furnace, together with 70 parts sodium bisulphate, 20 parts limestone, 22 parts sand and 7 parts coal. As soon as fusion takes place the material is tapped into a tank filled with water, which serves to granulate it. The material is then dried and ground and stored for market.

Manufacture of Phosphorus.

The Pelletier process for the manufacture of phosphorus by the conversion of tricalcium phosphate into monocalcium phosphate by means of sulphuric acid and the reduction of the product by charcoal has been studied by W. Hempel.[1] It was found that, to render the whole of the phosphate soluble, a large excess of sulphuric acid over that needed for the conversion of the tricalcium into the monocalcium salt must be used. If the excess of acid be neutralized by barium hydroxide, more free acid is formed on concentration, the monocalcium salt forming the dicalcium salt and free phosphoric acid. As the liquors containing sulphuric acid are concentrated, the amount of sulphuric acid retained diminishes, though not in inverse proportion to the concentration; and when a specific gravity of 1.28 is reached by concentration, corresponding with a sulphuric acid content of 4.2 per cent., phosphates begin to crystallize, so that further concentration is impracticable.

In the experimental work it was found that from 300 to 600 deg. C. all the sulphuric acid present was evolved as sulphur dioxide. At 700 deg. C. the evolved gas became combustible. At 740 deg. C. traces of phosphorus could be detected, but it did not come away in quantity till 960 deg., and at 1170 deg. C. the distillation was practically over. Early in the process, the gas evolved was a mixture of carbon dioxide, carbon monoxide and hydrogen; but the amounts of the first and last gradually lessen, and during the distillation of the phosphorus the accompanying gas consists almost entirely of carbon monoxide. There was accounted for in the condensing apparatus about 92.5 per cent. of the phophorus contained in the experimental mixture. An experiment with metaphosphoric acid showed that a good yield of phosphorus is not obtainable by reducing this substance with charcoal.

Wöhler long ago proposed to reduce a mixture of tricalcium phosphate and silica with charcoal, according to the equation $Ca_3P_2O_8+3SiO_2+5C=3CaSiO_3+5CO+2P$; but the process never succeeded, owing to the difficulty of attaining the requisite temperature or of finding vessels to withstand it. Electric heating, however, has lessened or removed these difficulties, and an experiment was made on this process. Gas began to be evolved at 700 deg. C., and became inflammable at 1000 deg. C.; phosphorus was

[1] *Zeitschrift für angewandte Chemie*, 1905, pp. 132-136.

detected at 1150 deg., began to distil in quantity at 1200 deg., and the distillation ended at 1450 deg. C. The yield was 92 per cent. (another experiment at higher temperatures gave no higher yield) and very little was in the "killed" condition. Electric heating has lately been adopted in German works. The retorts are sheet-iron cylinders lined with fire-clay, through the lower part of which carbon electrodes are introduced. The materials are fed in continuously, and the residual slag continuously removed. Perfect dryness of the materials is an important factor in increasing the yield.

PLATINUM.

There is a small production of platinum in the United States, amounting generally from 100 to 200 troy oz. per annum. It is believed that this production can be materially increased by proper working of the black sands of the Pacific slope, and the development of improved means for saving the small amount of platinum which much of this sand contains. An investigation for this purpose was conducted during 1905 in an elaborate way by the U. S. Geological Survey, at Portland, Ore., and has not yet been completed. Preliminary reports that have been officially published indicate a successful solution of the problem, but although an increased supply of platinum is to be expected from this source, it is hardly to be anticipated that the great shortage in the supply of this metal will be greatly relieved by the extraction from black sand.

A small amount of platinum has been obtained in previous years from the Rambler copper mine, Wyoming, but, so far as we are aware, there was no production from this source in 1905. The discovery of platinum on the plantation owned by Captain W. P. Love, near Love Spring, Cherokee county, South Carolina, has been reported, but the reports have not yet been verified. A reported discovery at Plymouth, Vt., is equally indefinite. Probably not much reliance is to be placed upon either report. Platinum is also known to exist in the valley of the Fraser river, in British Columbia, and some elaborate investigations upon this possible source of supply have been conducted recently by Howard W. DuBois, of Philadelphia, Penn. Although no specific statements have been made as to the results, it is inferred that they have been to some extent successful.

STATISTICS OF PLATINUM IN THE UNITED STATES.

Year.	Production. (a)		Imports.			Consumption.
			Unmanufactured.		Manufactured	
	Troy Oz.	*Value.*	*Troy Oz.*	*Value.*	*Value.*	*Value.*
1896	163	$944	83,080	$926,678	$106,338	$1,033,960
1897	150	900	83,080	960,299	43,921	1,005,120
1898	225	3,375	101,018	1,178,142	52,283	1,233,800
1899	300	1,800	187,778	1,462,157	55,753	1,539,710
1900	400	2,500	118,919	1,728,777	36,714	1,767,991
1901	1,408	27,526	85,438	1,673,713	24,482	1,725,721
1902	94	1,814	105,450	1,950,362	37,618	1,989,794
1903	110	2,080	114,521	1,921,772	135,889	2,059,741
1904	200	2,600	103,802	1,812,242	105,636	1,920,478
1905	200	(e)3,000	104,196	1,985,107	188,156	2,176,263

(a) Statistics of the U. S. Geological Survey. (e) Estimated.

The price for platinum continued high throughout 1905, starting at $19.50 for January and February, and remaining at $20.50 per troy oz.

for the rest of the year. Early in 1906 there was a further increase in value, the quotation rising to $25 per troy oz. The causes for this important movement in the market are primarily increased demand for almost all purposes, but especially in connection with the incandescent gas lighting industry, and shortage in the supply of crude metal delivered by the mines in Russia. The reasons for the latter condition are discussed further on in this article. The imports of platinum into the United States, chiefly crude metal which is refined here, are given in the table on p. 496, together with other statistics of the industry.

Outside of the United States, there is a small amount of platinum produced in Colombia, the output of which was 9625 troy ounces in 1904. The chief source of supply continues to be Russia, the statistics of which are given in the following table:

PLATINUM PRODUCTION OF RUSSIA. (*a*)

District.	1904.		1905.	
	Kilograms.	Troy Ounces.	Kilograms.	Troy Ounces.
Tcherdinsk	153.6	4,938	125.4	4,032
Perm	1,107.1	35,593	1,221.0	39,255
South Verkhotoorsk	3,538.5	113,763	3,536.9	113,711
North Verkhotoorsk	207.3	6,666	311.6	10,018
South Ekaterinburg	5.6	179	46.4	1,492
Total	5,012.1	161,139	5,241.3	168,508

(*a*) Privately communicated by W. A. Abegg, Warsaw. The total production in 1903 was 6003 kg. (193,000 oz.), and in 1902 it was 6133 kg. (197,173 oz.)

According to W. A. Dyes (*Chem. Ind.*, 1905, XXVIII, 387–381)[1], notwithstanding an increased supply in recent years, platinum has become considerably dearer, the price during 1904 ranging from 16,000 to 19,000 rubles per pood (1 ruble = $0.77; 1 pood = 36 ℔.). Both the price and production of Russian platinum have fluctuated considerably, as is shown in the following table:

Year.	Price.	Production.	Year.	Price.	Production.	Year.	Price.	Production.	Year.	Price.	Production.
	Rubles.	Poods.		Rubles.	Poods		Rubles.	Poods		Rubles.	Poods.
1869	1,600	140–0	1890	12,000	270–0	1893	....	311–3	1896	....	301–0
1874	....	120–0	1891	5,000	258–6	1894	....	318–0	1897	....	342–0
1882	....	250–0	1892	7,000	279–2	1895	....	369–5	1898	13,000	367–0

In 1900 the production was 310.7, and in 1901, 389 poods, while during the period of 1899-1904 it averaged 350 poods. The increased production of 1901 has not been maintained, owing chiefly to two reasons.

In the first place, the greater number of the more important mine-owners are bound by contracts covering a period of 10 years, whereby, whatever be the price of platinum in the markets of the world, they have to deliver their product to the large refining firms at a fixed price of 10,000 or 11,500

(1) Abstracted in *Engineering and Mining Journal*, August 26, 1905.

rubles per pood. These mine owners, not being in a position to receive any advantage from the rise in price of the metal, have, therefore, no inducement to increase the supply.

The second cause of the gradual decrease in the production of platinum lies in the diminution of the yield of the richer mines. In 1900 the maximum platinum content of the ore was 15.68 grams per 1000 kg., but in 1901 the corresponding figures were only 5.89 grams per 1000 kg. In 1900, 100 firms, and in 1901, 120 were engaged in platinum mining in Russia. The mines are all situated in the Government of Perm, in three districts, of which Gorablagodatj produced 193.4 poods in 1901 (118 in 1900), Krestowosdwischensk 99.8 poods (99 in 1900) and Mischnetagilsk 71.8 poods (76 in 1900).

A correspondent of the *Engineering and Mining Journal,* who has personal acquaintance with the conditions governing the Russian platinum industry, commented (Aug. 24, 1905) upon the above statements, remarking that the supply of platinum at the present time is indeed scarce, but, according to his information, the trouble has been chiefly the difficulty of securing workmen to produce the mineral. The Russian producers have been very much handicapped because of that fact, and inasmuch as the season ends practically Sept. 1, when the cold weather sets in, nothing can be done after that date until the following May. This is likely to cause the supply of platinum to be short during the winter of 1905–1906, and prices will likely be very high. After the troubles in the Far East are settled and during the summer months of 1906, a decrease in price to a more reasonable figure may be expected.

His prediction as to the course of the market during the winter just closed was certainly fulfilled. Late advices are to the effect that Russia proposes to restrict the export of platinum. The minister of trade and industry proposes shortly to summon a special conference to consider the questions of (1) the prohibition of the export of raw platinum, (2) the imposition of an export duty on platinum, and (3) the construction of Government works for refining platinum.

Among the publications relating to platinum during 1905 were a paper by E. Hussak, in *Oest. Zeit.*, of May 27, 1905, briefly describing deposits of platinum and palladium in Brazil; a paper by C. W. Dickson, on "The Distribution of the Platinum Metals in Other Sources than Placers," read before the Canadian Mining Institute, at the March meeting, 1905; a paper by R. Spring, in *Zeit. f. prak. Geol.*, February, 1905, entitled "Some Observations on the Platinum Washings of Mischnetagilsk," in which the occurrence, geological relations and mining of platinum in the Urals are described; and a very important paper on the "Assay of the Platinum Metals," by Dr. Nordenskjöld, in *Oest. Zeit.*, 1905, p. 473, and *Engineering and Mining Journal,* LXXX, p. 1017.

POTASSIUM SALTS.

By Reginald Meeks.

The Kali Syndicate continues to remain in its former strong position and to produce from 90 to 95 per cent. of the world's potassium salts, exclusive of the nitrate, or saltpeter. In February, 1906, a new agreement was signed to last three years. This supersedes that which expired at the end of 1905.

Some interesting developments have occurred relative to the entrance of American manufacturers into the German industry. The German Government has heretofore successfully kept the field free from outsiders seeking concessions for manufacturing, by limiting these on the ground of water contamination. This has been just, inasmuch as many of the streams in the neighborhood of Magdeburg and elsewhere have been rendered non-potable. A limit has been set upon the washing operations and the various watersheds are carefully watched. The number of new concessions therefore depends upon the alkalinity of the water of the streams. However, in former years, before the importance of limitation was realized, there were concessions granted which allowed unlimited washing facilities. The present condition calls for a production which amounts to about 100 tons per day per concession.

PRODUCTION OF POTASSIUM SALTS IN GERMANY. (a)
(In metric tons and dollars; 1 mark = $0.238.)

Year.	Kainite.		Potassium Chloride.		Potassium Sulphate.		Potassium Magnesium Sulphate.		Other Salts of Potassium.	
	Quantity.	Value. $	Quantity.	Value. $	Quantity.	Value. $	Quantity.	Value. $	Quantity.	Value. $
1896	856,290	2,989,736	174,515	5,718,559	19,682	813,381	4,623	85,977	902,707	2,964,750
1897	992,389	3,486,007	168,001	5,764,423	13,774	565,720	7,812	149,079	953,798	3,030,143
1898	1,103,643	3,835,856	191,347	6,380,220	18,853	763,397	13,982	259,485	1,105,212	3,576,628
1899	1,108,159	3,838,250	207,506	6,801,250	26,103	1,027,500	9,765	195,000	1,384,972	4,202,000
1900	1,178,527	4,134,000	271,512	8,793,750	33,853	1,249,250	15,368	280,500	1,874,346	5,643,750
1901	1,500,748	4,327,250	282,750	8,782,250	27,304	1,460,000	15,612	286,500	2,036,326	5,443,250
1902	1,322,633	4,571,980	267,512	7,507,710	28,279	1,079,092	18,147	334,390	1,962,384	4,949,448
1903	1,557,243	5,208,154	280,248	8,125,320	36,674	1,389,444	23,631	441,252	2,073,720	4,993,478
1904	1,905,893	6,322,470	297,238	8,425,676	43,959	1,664,572	29,285	545,972	2,179,471	5,305,972
1905	2,317,829	8,923,642	370,914	13,723,918	47,994	2,049,606	34,032	694,253	2,725,654	....

(a) *From Vierteljahrshefte zur Statistik des Deutschen Reichs*, except that the values for 1905 have been estimated by us on the basis of selling prices in the United States, less freight from the mines.

The Virginia-Carolina Chemical Company succeeded in purchasing the concession of one of the older companies and threatened to transport its crude product to the United States and wash its mineral here unless granted privileges nearer its mines. Thereupon the German Government conceived the idea of an export duty on crude mineral, which was somewhat in the nature of a club to use against the club of the invader. This unsettled and antagonistic state of affairs lasted until the American company

entered the syndicate, since which time everything has been amicable and the question of the threatened tariff has been laid aside.

It is well that such is the case, because the Government itself was dubious as to the proposed export tax, fearing tariff retaliation. The situation at present consists of a large increase in works (about 40, produced mainly by the Gamp law) and a considerable decrease in demand. It was therefore considered an unfavorable time to develop a measure of this kind.

The table on p. 499 gives the production of potash salts in Germany since 1896.

Potassium Nitrate.—About 20,000 tons of potassium nitrate are exported annually from India. The figures for export are so much in excess of those for production that the latter have practically no value. The following table shows the quantity of saltpeter exported from India during the last 10 years:

EXPORTS OF SALTPETER FROM INDIA. (a)
(In tons of 2000 lb.)

Year.	Quantity.	Value.	Value per 100 lb.	Year.	Quantity.	Value.	Value per 100 lb.
1895-6	23,610	$1,339,865	$2.83	1900-01	17,432	$1,471,245	$4.05
1896-7	29,583	1,430,410	2.42	1901-02	17,721	1,189,400	3.22
1897-8	20,889	1,329,155	3.06	1902-03	20,531	1,442,435	3.37
1898-9	18,263	1,164,480	3.06	1903-04	19,549	(b)1,470,000	(b)3.75
1899-00	19,870	1,281,050	3.64				

(a) From "Mineral Production of India," by T. H. Holland, Government geologist. (b) Value estimated.

In The Mineral Industry, vols. X and XI, will be found detailed accounts of the technology, geology and statistical data concerning the potassium industry.

The Refined Salts Market During 1905.[1]

Caustic Potash.—The demand was very active throughout the year; prices remained fairly stationary, despite competition, and quotations were from 4½ @ 5c. for old process and from ½ @ 1½c. more for new.

Carbonate.—The same condition existed in the carbonate market; opening prices for the year ruled at 3.65 @ 4.15c. for calcined and 4.15 @ 4.25c. for hydrated, depending on purity, make and quantity. Prices advanced and declined slightly from the above, finally closing steady at 3¾c. for calcined. Late in the year there were sales for future delivery, which brought 3¾ @ 4¼c. for calcined and 4⅛ @ 4⅝c. for hydrated.

Bitartrate.—Cream of tartar prices for the year remained at 23½c., except in January, when 24½c. was realized. The depreciation was due to lower values for raw materials.

Nitrate.—The market showed considerable strength throughout the year, the tendency being to higher prices and increased consumption. However-

[1]Taken mostly from the *Oil, Paint and Drug Reporter.*

ever, purchasers were loath to meet the increased prices. Opening quotations were 3.85c. for crude and $4\frac{1}{4}$c. for refined. Later, crude advanced to 4.55c., the high price, and for the remainder of the year fluctuations were confined to $\frac{1}{2}$c. Closing prices were $4\frac{3}{8}$c. for crude and $4\frac{1}{2}$c. for refined.

Chlorate.—There was not as much fluctuation in price during 1905 as in previous years. January opened firm at $8\frac{1}{4}$c. for crystals and $8\frac{1}{2}$c. for powdered, and these values remained steady up to the last quarter, when prices stiffened $\frac{1}{4}$c. owing to light stocks, closing for the year at $8\frac{1}{2}$c. and $8\frac{3}{4}$c. for crystals and powdered respectively. A year ago (1904) the high and low figures were $8\frac{1}{4}$ and $6\frac{1}{4}$c., while for 1903 the fluctuation was confined to between 7 and $6\frac{3}{4}$c.

Bromide.—This salt underwent the sharpest movement in price of any of the potassium products. The Kali Syndicate slashed prices from 30c. to 15c., thereby declaring war against its American competitors. Orders increased tenfold and many of these were considered speculative.

The cut in sodium bromide was even more marked. Prices recovered a few cents toward the close of the year, but all attempts at reconciliation were unavailing.

QUICKSILVER.

THE quicksilver product of the United States in 1906, as in the previous year, was derived from California and Texas, together with a small quantity produced at Mercur, Utah. The output of California was 24,600 flasks, of Texas 5000 flasks, and of Utah 1050 flasks. As noted in our last volume quicksilver is now marketed in flasks of 75 lb., instead of flasks of 76.5 lb., as formerly. The Black Butte district, of Oregon, made no production in 1905, but it is expected that the furnace which has been constructed there will be in operation in 1906. There is believed to be at this place a large deposit of low grade ore, which can be worked profitably by the improved methods which have been planned. The production of quicksilver in California and Texas is fully described in the articles by Messrs. Yale and Phillips, which follow this introduction. The statistics of the industry are given in the subjoined tables:

STATISTICS OF QUICKSILVER IN THE UNITED STATES.

Year.	Production.					Exports.			Imports.	
	Calif. (*a*) Flasks.	Texas. Flasks.	Others. Flasks.	Total. Metric Tons	Value.	Flasks.	Metric Tons.	Value.	Pounds.	Value.
1896.......	30,765	......	......	1,061	$1,075,449	19,944	692	$618,437	......	$2,037
1897.......	26,648	......	......	919	993,445	13,173	475	394,549	45,539	20,147
1898.......	31,092	(*b*)	153	1,077	1,194,746	12,830	445	440,587	81	51
1899.......	29,454	261	......	1,025	1,416,790	16,518	573	609,586	131	83
1900.......	26,317	1,700	233	974	1,279,436	10,172	353	425,812	2,616	1,051
1901.......	26,720	2,932	75	1,031	1,382,305	11,219	389	475,609	1,441	789
1902.......	29,552	5,252	......	1,208	1,515,714	13,247	459	575,099	*Nil*	......
1903.......	32,094	5,029	......	1,288	1,564,734	17,575	610	719,119	*Nil*	......
1904.......	28,876	5,336	700	(*c*)1,204	1,348,185	21,064	731	841,108	212	160
1905.......	24,600	5,000	1,050	1,043	1,140,750	13,460	458	497,470	2,690	1,710

(*a*) Reported by the California State Mining Bureau. (*b*) Included in "Other States." (*c*) Estimated; the weight of the flask was changed from 76.5 lb. to 75 lb. within this year.

At the present time the New Idria Quicksilver Mining Company, a Boston concern, is the largest single producer of quicksilver in the United States; its output in 1905 was 7200 flasks. This company is installing a third furnace, which will be completed in April, 1906. The Napa Consolidated Quicksilver Mining Company is probably the second largest producer, its present output being at the rate of about 400 flasks per month. The properties of the Boston, Ætna, Karl, and New Almaden companies have been abandoned.

There was a considerable increase in the quicksilver production of Utah in 1906, the total having been 1050 flasks, against 700 flasks in the previous year. The quicksilver output of Utah is derived from the mines at Mercur, where the occurrence of cinnabar has been known for many years (the

name of the district being indeed due to this fact), but the mines have been more valuable as gold mines than as quicksilver mines. The gold slime from the cyanide works at this place frequently shows quicksilver, dissolved by the cyanide solution, and precipitated by the zinc. Recently, ore of higher grade in mercury has been discovered, and a furnace for its treatment has been erected; it is said that the capacity of the plant will be increased.

Outside of the States above mentioned, quicksilver is known to exist only in Arizona, where cinnabar, disseminated through an earthy and silicious gangue, occurs in a vein at Cinnabar, 14 miles east of Ehrenberg on the Colorado river. The Colonial Mining Company, in 1905, sunk a shaft at that place, 180 ft. deep, on a vein 6 ft. wide, which is reported to show a good percentage of cinnabar. It is further reported that arrangements are being made to erect a reduction plant.

QUOTATIONS FOR QUICKSILVER IN LARGE LOTS—1903-1905.

Month.	1903.			1904.			1905.		
	New York.	San Francisco. Domestic.	San Francisco. Export.	New York.	San Francisco. Domestic.	San Francisco. Export.	New York.	San Francisco. Domestic.	San Francisco. Export.
January	$47.75	$45.50	$43.50	$45.75	$44.50	$41.50	$40.00	$40.42	$39.17
February	47.00	45.50	43.00	45.50	44.50	41.50	40.00	38.88	37.63
March	47.00	45.00	42.50	45.40	44.50	41.50	38.95	38.15	36.90
April	47.25	44.50	42.00	45.00	44.50	41.50	38.25	38.00	36.75
May	47.50	44.50	42.00	44.94	44.19	41.81	38.38	38.25	37.00
June	47.50	44.50	42.00	44.75	43.30	42.70	38.50	37.85	36.50
July	47.50	44.50	42.00	43.81	43.50	41.94	41.25	39.00	37.75
August	47.50	44.50	42.00	43.50	43.50	41.75	40.50	39.00	37.75
September	47.50	44.75	42.00	41.40	42.45	40.85	40.00	39.00	37.75
October	47.50	44.75	41.87	40.00	41.75	41.75	40.00	39.00	37.75
November	46.75	44.00	41.50	40.00	41.75	41.75	40.00	39.00	37.75
December	46.50	44.00	41.50	40.00	42.25	41.00	40.00	39.00	37.50
Year	$47.27	$44.67	$42.15	$43.34	$43.39	$41.63	$39.65	$38.80	$37.52

QUICKSILVER PRODUCTION OF THE WORLD.
(Metric tons.)

Year.	Austria.	Hungary.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1896	564	....	186	218	491	1,524	1,036	4,021
1897	532	....	192	294	616	1,728	965	4,327
1898	491	....	173	353	362	1,691	1,058	4,128
1899	536	....	205	324	360	1,357	993	3,775
1900	510	32	260	124	304	1,095	983	3,311
1901	525	33	278	128	368	754	1,031	3,120
1902	511	45	259	191	416	1,425	1,208	4,056
1903	523	44	314	188	362	914	1,288	3,633
1904	536	45	357	(e) 190	393	1,020	1,192	3,733
1905	519	(e) 45	370	(e) 190	318	800	1,043	3,285

(*e*) The figures for Austria, Italy and Spain for recent years have been furnished by V. Spirek. (*e*) Estimated.

Quicksilver Mining in California.

By Charles G. Yale.

Affairs in the quicksilver-mining industry in California were in rather a bad way in 1905—worse, in fact, than they ever have been. Prices were low; there was an over-production and a decided decrease in consumption. There was a marked decrease in production in 1905, due in a large part to

the lack of demand at profitable rates. The smaller mines can not sustain themselves at present prices of the metal, and even the larger ones are making only small profits. Where the mines are fully developed and equipped, where fuel is cheap, and where the orebodies are within easy access to reduction works, they can continue to work; but otherwise some of them will have to be closed down.

The consumption of quicksilver in the mining districts of the United States has fallen off materially in late years, notwithstanding the expansion of the mining industry and the rapid increase in the number of properties being worked. The conditions of ore treatment have greatly altered; much more ore is now smelted than was formerly the case. Very few silver mines are now being worked where the pan-amalgamation process is in use. In the bonanza days of the Comstock the loss of mercury per ton of ore was about 7 ℔. Naturally, immense quantities of quicksilver were used. In working gold ores, by the ordinary milling process of the present, the loss of mercury is small.

Of late, China, and particularly Japan, have been the largest consumers of quicksilver, drawing supplies from the California mines. The Japanese, who have been using an explosive (the composition of which is kept secret), have been giving unusually heavy orders for mercury from California; probably fulminate of mercury has entered to some considerable extent into the manufacture of the explosive. The mercury which goes to China is made into vermilion. This Oriental demand has been a good thing for the California quicksilver miners, who otherwise would have had even less chance to sell their product. At the same time quicksilver sold in the Orient has to compete with that derived from Spain, Austria and Italy; it must therefore be sold at a lower price than in the United States, where the metal is protected by a duty of 7c. per ℔. In fact, this duty has been the salvation of the quicksilver miners of this country. But when we sell to China and Japan, we have to dispose of the product at from $5.25 to $7 per flask less than we obtain for it in this country.

It is for this reason that the quicksilver mine owners, instead of selling their product independently, dispose of it through selling agencies. The quicksilver is stored in San Francisco; all that can be sold is sold in the United States, at the best prices possible. When stocks become large and must be sold, certain portions are sent to foreign countries, such as Japan and China, at lower prices. By having all the supply of quicksilver "pooled" each mine, in proportion to its output, gets its proportion of both home and foreign sales, according to price.

If this system were not pursued, some mines would get all the advantage of the high prices of home sales; and others would have the disadvantage of foreign sales at low prices. Competition in this respect would break the already bad market, and would reduce the present prices.

Of course the miners would not export at all if there were a good market and demand in this country. Therefore the export prices depend, to some extent, on the surplus production in this country. As already stated, there is a difference of from $5.25 to $7 per flask between the price obtained here and that obtained in Japan.

The average price obtained for California quicksilver in New York during 1905 has been about $38.50 per flask of 75 ℔.; it has been about 50c. per flask lower in this State. This figure does *not* consider the export prices, which, as stated, have been lower. Therefore, with the large quantity exported, the quicksilver miners of the State have not really received for their product the average value quoted.

The grade of cinnabar ore now being worked in California is much lower than formerly; seemingly, the high-grade ore deposits have been exhausted, except where small bunches are occasionally found. Many mines are now working on ore carrying 0.5 per cent. metal; only those worked on a large scale and with full equipment can maintain themselves or make a profit under present conditions. The quicksilver mines in Texas have a higher grade of ore, some of them handling 2.5 per cent. ore; they are doing well. But ore of 1 per cent. is now considered high grade in California.

As an indication of the decadence of quicksilver mining in this State, the once famous New Almaden mine of Santa Clara county (owned by the Quicksilver Mining Company of New York) is about to close down. This mine alone has produced far more quicksilver than all other mines of the United States put together; it was worked before gold mining commenced in California and has been in operation ever since. For the last few years it has been working over old dumps, and such ore as it was able to get in the mine; but it is understood that there has been no profit. During one year of the life of this mine, for 12 consecutive months, the average output of all the ore was 36 per cent. quicksilver; even then the furnaces were badly constructed and leaky, so that there must have been considerable loss in reduction. In its prime, this famous mine produced much more quicksilver in one month than it has done of late in a year.

The outlook for quicksilver mining in California is discouraging; no one is able to predict confidently any material change for the better within the immediate future. No prospective increase in demand is apparent. The present conditions threaten the closing down of a number of smaller mines in this State; it is improbable that many new ones will be opened and developed, unless the ore should happen to be of unusually high grade.

Several quicksilver properties in California have been under development in the last few years. The following list comprises those which have been producing more or less during 1904 and 1905: Colusa county, the Manzanita. Lake county, the Abbott, Chicago, Great Western, Helen, Mirabel, and Boston (or Redington). Napa county, the Corona (or Vallejo)

Etna, Johnson, Manhattan, Napa Consolidated, Twin Peaks, and Wall Street. San Benito county, the New Idria (largest producer in the State) and Ramirez. San Luis Obispo county, the Alice and Modoc (or Little Bonanza Company), Oceanic, Polar Star, Clark, Karl and Riconada. Santa Clara county, the Century (or Guadalupe), and New Almaden (Quicksilver Mining Company). Solano county, the St. John, Sonoma county, the Cloverdale, Culver-Baer, Great Eastern, Mount Jackson, Socrates, and Sonoma. Trinity county, the Altoona.

The Quicksilver Industry in Brewster County, Texas.

By William B. Phillips.

During 1905, important results followed close upon developments, both in the limestones of the lower and the shales of the upper Cretaceous. In addition to the operations of the Marfa & Mariposa Mining Company, now, as heretofore, the largest producer in the district, with its two 10-ton Scott furnaces, the Lone Star Mining Company carried forward considerable work in the limestone area, a mile east of Terlingua post-office. The same general conditions have attended this, as all similar work in these rocks. The pockets of furnace material are sometimes connected by means of stringers which may or may not carry ore. While the general trend of the orebodies is towards the northeast, there are many and notable instances of deposits which appear to follow no rule of strike. In the limestone area, successful work consists largely in prospecting for new pockets to take the place of those that have been worked out. For several years the Marfa & Mariposa Mining Company confined its operations to deposits near the surface, the greater part of the work consisting of pits, trenches and shallow shafts. Within the last two years, however, this company has explored its ground in depth and one of its shafts is below the 300-ft. level. This company has also worked a good deal of ground near the contact of the limestones with an igneous flow, andesitic in character. Excellent ore has been found in this association.

The Lone Star Mining Company and the Terlingua Mining Company have also been working in the Edwards and Washita limestones. The former company put in 12 Johnston pipe retorts and has been pleased with the results. These retorts are arranged in one horizontal plane and are all heated from one fire-box. Each retort holds from 200 to 250 lb. of ore, and the fumes from each retort are delivered into a condenser pipe, of cast iron, 8 ft. long and 3 in. in diameter. There is a cemented trough into which all of the condenser pipes empty and the metal flows into a receptacle by gravity. Theoretically one set of 12 retorts should treat 4800 lb. of ore in 24 hours, allowing 200 lb. for each retort. By this calculation each retort is charged and emptied twice in the 24 hours. In practice, however, and with the class of material that has been sent to them, it has not been pos-

sible to treat more than 4000 lb. per 24 hours. The consumption of fuel is very low, not over $\frac{1}{4}$ cord per 24 hours, and the labor cost is also low, two men on the day shift and one man on the night shift being sufficient. The writer had the opportunity of testing these retorts on a run with 9000 lb. of bituminous ore from the Eagle Ford shales. The ore carried 1.74 per cent. of quicksilver and the actual recovery of metal was 81 per cent. of the assay value. This was a lower yield than has been obtained with an ore not so bituminous in character, for the oily nature of the distillate rendered it difficult to clean the pipes thoroughly and some metal was left adhering to the inside.

It is often observed in distilling bituminous ore in a retort that there is formed in the pipes a hard crust rich in metal and difficult to remove. A semi-circular stiff steel brush would probably obviate this to a great extent, or a boiler-flue cleaner of the expanding type. The Johnston pipe furnace seems to be well adapted to the treatment of soot from a Scott furnace, although it has also done good work on ore. The cost of 12 Johnston retorts in the Terlingua district, set up and ready to run, varies from $1300 to $1400. In connection with prospecting work, where it is not necessary to treat large amounts of material and where operating expenses have to be kept down to the lowest possible limit, the Johnston furnace is to be recommended. Successful experience with this furnace in California justifies this opinion.

During the summer of 1905 the Terlingua Mining Company made a run on ore that had been accumulated for the purpose. This company has a 50-ton Scott furnace. It is not run continuously, however, the policy of the company being to mine ore during the fall, winter and spring and operate the furnace during the summer. A successful run was made last summer on the usual class of ore obtained from the limestone area.

The Colquitt-Tigner 10-ton Scott furnace, on Section 38, Block G 12, has not been in operation for two years. It has recently been leased by the Chisos Mining Company and is now in commission again. The equipment of the companies in the limestone area is as follows: Marfa & Mariposa Mining Company, two 10-ton Scott furnaces; Terlingua Mining Company, one 50-ton Scott furnace; Colquitt-Tigner Quicksilver Mining Company, one 10-ton Scott furnace; Lone Star Mining Company, one bench (12) Johnston pipe furnace. The daily capacity of the furnaces in this area is thus 82 tons of ore a day, but continuous operations have been carried on only by the Marfa & Mariposa Mining Company, with two 10-ton furnaces.

The limestone area proper is about five miles east and west and two miles north and south. These limits comprise all of the successful work that has been done during the last several years. The thickness of the ore-carrying limestones may be taken at about 1600 ft., but they have not been

explored much below the 300-ft. level. Nearly all of the quicksilver produced in Brewster county has been yielded by the limestones of the lower Cretaceous (Edwards, Fredericksburg, Washita, etc.). This particular area is confined between Fresno Cañon on the west and Vogel's draw (Long draw, of the United States Geological Survey maps) on the west.

The elevation of this district, which should be known as the Terlingua district, is about 3000 ft., the drainage beng to the south and southeast. It is much dissected by deep cañons, one of them, Crœsus Cañon, attaining a depth of nearly 700 ft. The country rock throughout this area, with a few exceptions on the east side, is a heavy, close-grained and dense limestone traversed by veins of calcspar and gypsum and characterized by numerous caverns, some of which do not appear from the surface. Some of these caverns are lined with crystals of calcspar and some with most curious and beautiful growths of gypsum. Occasionally, as on Section 38, Block G 12, these caverns afford a considerable amount of fine ore, found in piles on the floor as if placed there by human agency. Many, if not all of these caverns, are old water-ways, and one of them is still lined with calcspar coated with cinnabar. While this area has been classed as Lower Cretaceous yet there has been discovered in a cavern the remains of what may prove to be a sloth of later age than Lower Cretaceous. It may be that the animal, living in a later age, was imbedded in a swamp or depression which afterwards was covered over by the limestone. Be this as it may, there can be no question that the age of the limestones holding the ore is Lower Cretaceous.

Three miles east of Terlingua post-office is a notable "draw," or long, wide cañon, known as Vogel's draw. It marks, in a general way, the eastern limit of the Lower Cretaceous. Its western edge is a high and steep escarpment of limestone, the shales and thin bedded flags and limestones of the Upper Cretaceous coming in on the east side and continuing for several miles. In this area the mining conditions are radically different from those which maintain in the Terlingua district on the west. The name Study Butte district is proposed for this area. It is characterized by the gray and bluish-black shales of the Eagle Ford (Upper Cretaceous). These shales are held between two beds of gray sandstone, each bed carrying coal measures and being productive of a fair quality of coal. The general dip of these shales and of the sandstone is towards the south and east at angles varying from 16 to 25 deg. In going towards the south from Cigar Springs towards Needle Peak one passes directly from the coal-bearing sandstone into the cinnabar-bearing shales, and then into the coal-bearing sandstones again, all with the normal dip as mentioned.

In the development of the Study Butte district this is an important consideration. Two of the largest companies that have operated in the district have used the coal as fuel under steam boilers with good results.

The coal has also been used under the retorts which have been operated by the Chisos Mining Company. The increasing scarcity of wood brings up for consideration the question of making gas from this coal and using it as fuel under boilers, in gas engines and in the furnaces. It is idle to assume that the only fuel suitable for quicksilver furnaces is wood. In California oil is used, and there is no good reason why producer gas may not be used with marked economy. The time may not be so very far distant when the local coal will be used for the production of gaseous fuel, and this fuel will be used in gas engines for driving machinery, and in the furnaces for distilling quicksilver. This question is already under serious consideration by some of the companies in the Study Butte district. Good mesquite wood is now held at about $7 per cord, delivered at the furnace. Good coal is quoted at $5 to $6.50 per ton, delivered, and it is entirely feasible to reduce this price by $1 or $2 per ton. Some of the coal makes fair coke, although lacking in strength. The workable coalfields in the southern part of Brewster county are worth a closer examination than has been yet given to them. The coal measures occur over a larger area than has been anticipated and the quality of the coal, for steaming purposes, is much above the average of Texas coals. The center of the field, however, is about 80 miles from rail, the nearest point being Marathon, on the Southern Pacific.

In the Study Butte district are the following companies: Texas Almaden Mining Company, one 20-ton Scott furnace; Big Bend Cinnabar Mining Company, one 50-ton Scott furnace; the Chisos Mining Company, four D-retorts. This company has recently leased the 10-ton Scott furnace of the Colquitt-Tigner Quicksilver Mining Company, on Sec. 38, Block G 12.

The Texas Almaden Mining Company has just built its furnace and expects to put it in commission during the spring of 1906. The furnace of the Big Bend Cinnabar Mining Company has been idle since September, 1905. These two companies are mining on the eastern side of the Study Butte district about a mile east of Terlingua creek. For the most part, the ore at this place is held in rhyolite, which occurs as a capping of more than 200 ft. thickness over the shales. In the rhyolite the cinnabar occurs as stringers, varying in thickness from 0.25 in. to 3 and 4 in. These stringers penetrate the rhyolite in all directions and are often closely associated with pyrite. There is much water in the mines on the eastern side of Study Butte district and heavy pumps are required to handle it. Steam boilers are used, the water being pumped from Terlingua creek. These two companies are the only ones in the entire quicksilver field that are using steam; gasoline engines are used at all the other places.

The chief producer in this district, up to this time, is the Chisos Mining Company, just east of Vogel's draw. Operations have here been conducted for the last three years, the D-retort being used. This retort holds

from 750 to 800 lb. of ore and is heated as a gas retort. It has not been found profitable to send to these retorts ore with less than 4 per cent. of metal, and consequently the material mined has been hand-sorted. The consumption of wood for two retorts, built one above the other, has been 1.5 cord per day of 24 hours. From 40 to 60 lb. of lime is added to each charge, the lesser amount when the ore is dry. No water has been encountered in the workings down to the 180-ft. level, but during the rainy season the ore piles become wet and additional lime is used. This company has leased the 10-ton furnace of the Colquitt-Tigner Quicksilver Mining Company and is now hauling ore to it, five miles.

Throughout the entire quicksilver area, with the exception of that portion lying east of Terlingua creek, the scarcity of water necessitates the construction of tanks and the hauling of water from the Rio Grande and from tinajas and springs. For a 350-gal. tank the ordinary price is $3.50, or one cent a gallon. A great deal of water is hauled from the Rio Grande, 10 miles and up a grade of nearly 1000 ft. The current system of supplying the men with water is to allow one bucket for each single man per day and from two to five buckets for a family. The rainy season begins about the middle of June and lasts until September, the annual fall being about 12 in. During the summer months the mid-day temperature may reach 114 in the shade and from 125 to 135 in the sun. During the winter the temperature may fall as low as 25 deg. F., but this is exceptional. From time to time there may be little flurries of snow, but there is scarcely a day when out-door work may not be carried on without discomfort.

There is little or no vegetation in the field except that of the ordinary desert type, palma, palmilla, oquitilla, lecheguilla, tasajilla, croton-weed, popotilla, many varieties of cactus, junco, etc. Of shade there is none save what one makes for himself. The near-by supply of wood for the furnaces is becoming scarcer, and this has induced some of the companies to inquire into the feasibility of using the local coal in gas producers. It may be a little early in the history of the field for this change to be made, but that it will come in the course of time the writer does not doubt.

There is little or no outside prospecting for quicksilver. The State mining laws are practically prohibitory of prospecting, the certainty of the expense and the uncertainty of the locations deterring those who might otherwise feel sufficient interest in the matter to look more closely into it. The public lands are held as a sacred trust for children whose great-grandparents are yet unborn. In the mean time, prospectors and mining men are pouring into New Mexico, Arizona and old Mexico.

The price of labor in the field is as follows: Blacksmiths and carpenters, $3 to $4 per day; miners (barrateros), $1.25; muckers (resagadores), $1; whim tenders, $1.25; stone masons, $1.50; machine-drill men, $1.50; engine- and firemen, $2 to $3. Nearly all of the labor is Mexican, the really

responsible positions being filled by Americans. Power drills are not much used, one of the companies (Marfa & Mariposa) having an air compressor and air drills. The Chisos Mining Company, having softer ground, uses the Jackson hand-power drill.

The future of the Texas quicksilver industry will depend more and more on the ability of the companies to mine and treat ores of medium grade—1 or 2 per cent. While rich pockets are often met with, especially in the limestone area, yet no great amount of business can be based on them. They are not frequent or continuous enough to allow any reliable estimates. Considerable quantities of rich ore are often secured from them, but for regular operations they may as well be put to one side. It seems to the writer, after an experience in this field for the last five years, that in deep mining lies the hope of the industry. Not deep mining in the sense in which this expression is employed on the Rand, for instance, but deep as compared with anything that has yet been done here. This is particularly true of the deposits in the bituminous shales of the Eagle Ford (Upper Cretaceous), and the companies operating in this formation might as well make up their minds that the main bodies of ore are below the 300-ft. level and probably below the 500-ft. Nearly all of the metal obtained from the field has come from comparatively shallow depths, not exceeding 100 ft., but this argues for nothing more than the well-known fact of the occurrence of rich ore near the surface. It does not affect the main question at all,whether the workable ores that are to be depended on for further supplies are not below the present workings. The thickness of the Eagle Ford shales may be taken as 800 ft., but they have not been explored beyond a depth of 180 ft.

The plans for the farther development of the property of the Chisos Mining Company have taken this into consideration. No richer ore has been found near the surface than has been found on the 150-ft. level. The ground is firmer, the veins hold a steadier course, and the distribution of the ore in the vein of calcspar is more uniform.

Furthermore, the bituminous matter in the ore acts as fuel in the furnace and diminishes the amount of wood that has to be consumed. The bituminous ore is not met with near the surface, except where erosion has cut away the overlying gray shales and flags. It is at present impossible to say whether these bituminous shales carry commercial amounts of oil in depth or not. No bore-holes have been put down and the matter is left to conjecture. On distilling the bituminous shales in a D-retort there are obtained three products, quicksilver, a lubricating oil and illuminating gas. It is possible to work at night by means of the light given off on igniting this gas as it comes from the condenser pipes. Lubricating oil, suitable for use on common machinery, is obtained from the condenser pipes together with the quicksilver.

SALT.

The production of salt in the United States amounts to between 20,000,000 and 25,000,000 bbl. per annum, there not having been during the last seven years any decided tendency toward increasing production. In 1899—1901 the output was about stationary at 20,000,000 to 21,000,000 bbl. In 1902 it jumped to nearly 24,000,000 bbl., but in 1903 fell off to 19,000,000 barrels.

Salt is produced in New York, Michigan, Ohio, Kansas, Louisiana, California, West Virginia, Texas, Utah, and a few other States, in which the output is relatively insignificant. The States mentioned by name ranked in the order given, until 1905, when Michigan first surpassed New York as a salt producer.

PRODUCTION OF SALT IN THE UNITED STATES.

(In barrels of 280 lb.) (a)

Year.	California.	Illinois.	Kansas.	Louisiana.	Michigan (c)	Nevada.	New York. (c)	Ohio, W. Virginia and Pa.	Utah.	Other States.	Total Barrels.
1900..	455,271	55,000	2,350,000	386,744	6,845,685	5,786	8,123,550	1,688,286	267,857	560,550	20,738,729
1901..	601,659	99,700	2,087,791	451,430	7,729,641	13,781	7,286,320	1,385,257	334,484	569,092	20,566,661
1902..	682,680	90,009	2,158,486	399,163	8,131,781	14,829	8,523,389	2,318,579	417,501	1,112,824	23,849,221
1903..	629,701	(d)	1,555,934	568,936	4,297,542	(d)	8,170,648	3,043,135	212,995	489,238	18,968,089
1904..	821,557	(d)	2,161,819	1,095,850	5,425,904	(d)	8,600,656	3,030,829	253,829	639,558	22,030,002
1905..	664,099	(d)	2,098,585	1,055,186	9,492,173	(d)	8,359,121	2,728,709	177,342	1,390,907	25,966,122

(a) Statistics of the United States Geological Survey. (c) Includes brine used in manufacture of alkali. (d) Included in "Other States."

SALT PRODUCTION OF THE CHIEF COUNTRIES OF THE WORLD.

(In metric tons.)

Year.	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Algeria......	19,658	23,222	21,300	17,378	18,325	18,518	27,263	26,329	18,563	(c)
Austria.....	308,932	331,084	341,959	342,059	330,277	333,238	310,807	359,014	369,875	(c)
Canada.....	39,880	46,584	51,853	53,847	56,296	53,927	57,203	56,671	62,411	41,170
France.....	1,042,614	948,000	999,283	1,193,532	1,088,634	910,000	863,927	967,531	1,153,754	(c)
Germany...	1,303,319	1,306,684	1,370,341	1,432,181	1,514,027	1,563,811	1,583,458	1,693,935	1,701,654	1,777,535
Greece......	22,800	20,421	37,125	22,411	22,411	23,079	25,200	26,000	27,000	(c)
Hungary....	180,133	171,711	178,551	182,593	189,363	(a)211,321	174,882	183,327	187,620	(c)
India (d)....	1,043,172	937,888	1,043,828	977,240	1,021,426	1,120,187	1,056,899	908,911	1,188,900	(c)
Italy.......	29,274	31,526	29,745	28,842	367,255	435,187	458,497	488,506	464,326	(c)
Japan.......	531,909	621,731	646,719	390,433	669,694	659,118	620,820	657,489	(c)	(c)
Russia......	1,346,247	1,551,894	1,505,600	1,681,362	1,768,005	1,705,922	1,847,019	(c)	(c)	(c)
Spain.......	521,751	508,606	479,358	598,108	450,041	345,063	426,434	427,394	543,674	551,900
U. Kingdom.	2,054,715	1,933,949	1,908,723	1,945,531	1,873,601	1,812,180	1,924,273	1,917,184	1,921,899	1,920,149
U. States....	1,995,017	2,009,625	2,382,197	2,522,610	2,651,278	2,612,204	2,409,174	2,408,646	2,797,461	3,297,285

(a) Sales by the royal monopole, including imports entered for consumption. (c) Statistics not yet published. (d) Does not include the untaxed output of certain native States.

Output of additional countries during 1904, in metric tons: Australia, 42,737; Bahamas, 1,595; Ceylon, 36,488; Cyprus, 3,954; Turks' and Caicos Islands, 30,951; Peru, 18,544; Tunis, 23,600; Venezuela, 7,410 tons.

The output in 1905 was constituted by the following grades: Table and dairy, 2,380,808; common fine, 6,818,690; common coarse, 3,140,384; packer's, 327,192; solar, 487,528; rock, 4,733,765; other grades, 207,824; brine, 7,869,931 barrels.

Imports and Exports.—The salt imported and entered for consumption during 1905 amounted to 320,421,880 lb.[1] ($492,031) compared with 330,933,483 lb. ($516,828) in 1904. Exports of domestic salt amounted to 68,475,356 lb. ($239,223) in 1905 against 27,978,090 lb. ($113,625) in 1904. The amount of foreign salt re-exported in 1905 was 611,912 lb. ($893) and in 1904 it was 2,089,234 lb. ($2,814).

[1]Figures of the Bureau of Statistics. Obtained by adding to (or subtracting from) the annual imports the decrease (or increase) in the amount of salt remaining in bonded warehouses between the beginning and end of the year.

SODIUM AND SODA SALTS.

By Reginald Meeks.

There is a considerable production of metallic sodium, which is not, however, marketed largely in metallic form, but is used as the basis of manufacture of certain soda salts, especially the cyanide. The Niagara Electro-Chemical Company (a branch of the Roessler & Hasslacher Chemical Company) at Niagara Falls, is a large producer of sodium for this purpose. The production of metallic sodium in the United States in 1905 has been estimated by a good authority at 1200 tons, the nominal value being 60c. per lb. or $1200 per ton. The same authority estimates the production of sodium in Great Britain at 1200 tons, and in Germany at 1000 tons; giving a total production of 3400 tons for the world. Further details as to this industry are not available.

During 1905 there were no new developments of unusual character in the soda industry. As in former years the trade in nitrate of soda held a prominent position, though production continues to advance only to a limited extent, owing to the agreement among the Chilean *oficinas*.

The world's production in 1905 amounted to 1,733,000 long tons, of which 321,231 tons were imported into the United States, the remainder being absorbed by European and other countries. This compares with a consumption of 1,447,000 tons in the previous year and 1,412,000 tons in 1903.

The Chilean nitrate industry is of the greatest importance to that Government, because an annual revenue of something like $20,000,000 in gold is derived from an export duty of $12.32 per ton on the salts.

There are four producing districts in the nitrate area of Chile; these contribute in the following proportion: Tarapacá, 81 per cent.; Tocopilla, 9 per cent.; Taltal, 8 per cent. and Antofagasta 2 per cent. There are 80 *oficinas* in the field, of which 68 are in Tarapacá. The industry furnishes employment for about 25,000 men. It is worthy of notice that in all the enormous output of nitrate of soda in northern Chile there is no American capital interested, although shipments to the United States are increasing. The companies are mostly English, with a few Chilean and German.

The cost to consumers has increased in the last two years owing to higher rates for freight and wages, and curtailed production. That the industry has been and is profitable may be seen by the following annual report of the London Nitrate Company, Ltd., Oct. 31, 1905. The company was organized in 1887 with a capital of $800,000. In 18 years

it paid in dividends $1,170,000 and returned to its stockholders $400,000 of its original capital, making a total of $1,570,000 cash. Besides this the company established new plants at a cost of $400,000 and began a sinking fund invested in English securities of $100,000 more. This shows that the total cash earnings in the 18 years had reached the sum of $2,070,000. It still has on hand a capital stock of $400,000 and assets worth, at low valuation, $355,000.

It has been claimed that valuable deposits of sodium nitrate exist in San Bernardino county, California. Some of these reports are of a very sanguine nature and even claim that the supply may be sufficient for the United States.

From reliable sources it is learned that the alleged deposits of California contain practically no nitrate of any commercial value. A few samples showed traces of the salt, but the majority were entirely devoid of even traces. It is significant that three of the largest nitrate concerns in this country have investigated the field and have decided that there is no commercial value attached to the discovery.

The United States is a large consumer of Chile saltpeter and imports close to one-quarter of the world's supply. The annual imports and values for the last ten years are shown in the subjoined table:

IMPORTS OF SODIUM NITRATE INTO THE UNITED STATES. (*a*).
(In tons of 2240 lb.)

Year.	Quantity.	Value.	Value. per ton.	Year.	Quantity.	Value.	Value per ton.
1896	115,504	$3,566,744	$30.88	1901	208,679	$5,999,098	$28.75
1897	94,965	2,810,187	29.59	1902	205,245	5,996,205	29.21
1898	147,495	2,298,240	15,58	1903	272,947	8,700,806	31.88
1899	146,492	3,486,313	23.80	1904	228,012	9,333,613	40.93
1900	182,108	4,935,520	27.10	1905	321,231	11,206,548	34.89

(*a*) From *Summary of Commerce and Finance of the United States.*

Natural Soda.—A deposit of soda is located at the Carisa Plains dry salt lake, 13 miles from Olig, in the eastern part of San Luis Obispo county, California. The deposit is in a dry lake six miles long by one mile wide, and varies in depth from a few inches to several feet. The underlying water is said to be highly charged with soda; in the winter it flows into the basin and in the summer evaporates, leaving crystals of soda behind. The Carisa Chemical Company of Bakersfield, California, is utilizing the deposit and has installed the necessary apparatus.

The Markets in 1905.

January opened with prices for nitrate between $2.35 and $2.40. These shaded off in the two following months only to stiffen considerably in April, the price advancing to $2.45 @ 2.52½, and again in May, when $2.77½ was the high price for the month and year. June saw a drop of 30c., followed by an advance of 5c., then prices receded and remained

at \$2.22½ @ 2.20 for the remainder of the year. The high and low prices for 1905 were \$2.77½ @ 2.20, as against \$2.37½ @ 2.10 for the preceding year.

Prices for bicarbonate remained practically stationary, opening at \$1.30 for bulk and \$1.50 in kegs f.o.b. works. The demand for caustic soda was very active during the first quarter of the year, and large sales of domestic were made, deliveries to take place in 1905, 1906 and 1907. Quotations were on the basis of \$1.85 for 60 per cent. The demand remained strong throughout the year and prices did not vary. Salt cake was held firm at 65c., with the demand limited. The deliveries were principally confined to contracts. Sharp competition existed among the domestic manufacturers of sal soda, and prices fluctuated. In April prices advanced sharply, and imported was sold freely at the advanced price of 85c. The domestic article was quoted at 60c. f.o.b. works or 70c. New York. Bleaching powder underwent a wide variation in price. January quotations were at \$1.75, and for the first quarter this price varied only 10c. The high prices for each month remained almost constant, but low prices were variable, touching \$1.25 and \$1.00 during the last quarter.

SULPHUR AND PYRITES.

By Edward K. Judd.

During 1905, the Union Sulphur Company, of Louisiana, became still more strongly intrenched in its command of the domestic market. It may almost be said that it is only through the tolerance of this company that any Sicilian sulphur now finds its way into the United States. The Union Sulphur Company, for commercial reasons, is not at present willing to report its output, wherefore exact statistics of the production of sulphur in the United States, either in 1904 or 1905, cannot be given. During 1905 the company did, however, enlarge its capacity, and must undoubtedly have exceeded greatly its output of the previous year. Assuming that the domestic consumption of sulphur in 1905 held to the same rate of increase that has been recorded in late years, and comparing with the imports of brimstone and the known production of pyrites, we estimate the shipments from Louisiana in 1905 to have been 228,000 long tons. We can say nothing as to production in that year, but it is known that stocks on hand, ready for shipment at its end, totaled several hundred thousand tons. The small output of sulphur in Utah and Nevada, all of which is consumed on the Pacific coast, was about the same as in the previous year, and is estimated at 4000 tons.

CONSUMPTION OF SULPHUR IN THE UNITED STATES.
(In tons of 2240 lb.)

Source.	1901.	1902.	1903.	1904.	1905.
Sulphur—Domestic production	6,866	7,443	35,098	193,492	232,000
Imports	175,310	176,951	190,931	130,421	84,579
Total	182,176	184,304	226,029	323,913	316,579
Exports	207	1,253	967	2,493	1,713
Consumption	181,969	183,141	225,062	321,420	314,866
(a) Sulphur contents at 98 per cent	178,330	179,478	220,560	314,992	308,569
Pyrite—Domestic production	234,825	228,198	199,387	173,221	200,280
Imports	403,706	440,363	427,319	413,585	515,722
Total	638,531	668,561	626,706	586,806	716,002
Exports	......	3,060	1,330	......	......
Consumption	638,531	665,501	625,376	586,806	716,002
Sulphur in domestic at 44 per cent	103,323	104,071	87,730	76,217	88,123
Sulphur in foreign at 47 per cent	189,742	205,532	200,215	194,385	242,389
Total sulphur content	189,742	309.603	287,945	270,602	330,512
Grand total sulphur consumption	471,395	489,081	508,505	585,594	639,071

(a) Includes crude and refined sulphur.

The production of pyrites in 1905, exclusive of the western pyritic gold-silver ores, but including some pyrrhotite mined for sulphuric acid in

Virginia, as well as some marcasite recovered by the lead and zinc mills of Wisconsin, amounted to 200,280 long tons, valued at $651,796, as against 173,221 tons, worth $669,124, in 1904. The increase was most pronounced in Virginia, which contributes nearly three-quarters of the entire output, and in New York, where two new companies in St. Lawrence county have lately become active producers.

SULPHUR.

Louisiana.—The sulphur beds of Calcasieu parish, whose heavy overburden of quicksand and water-bearing strata baffled for many years all attempts at shaft sinking, became available only when attacked by the Frasch system. This has been fully described in previous volumes. The sulphur-bearing horizon ranges from 150 to 250 ft. thick, and the total amount of sulphur contained within the area now owned by the Union Sulphur Company has been estimated at 40,000,000 tons, but it is uncertain how much of this will be extractable.

Sicily.—Shipments of sulphur from Sicily fell in 1905 to their smallest amount since 1898; stocks on hand at Sicilian ports at the end of 1905, on the contrary, were larger than ever before.

TOTAL EXPORTS OF SULPHUR FROM SICILY, 1900–1905. (*a*)
(In tons of 2240 lb.)

Country.	1900.	1901.	1902.	1903.	1904.	1905.
Austria	21,594	18,842	19,086	17,926	23,374	25,111
Belgium	9,721	7,471	12,323	15,233	13,627	14,442
France	103,647	74,394	67,249	74,372	103,040	96,170
Germany	28,702	23,448	25,906	32,553	31,613	28,319
Greece and Turkey	19,647	21,702	20,548	22.133	25,376	25,069
Holland	18,595	10,848	8,648	5,157	8,122	4,425
Italy	101,073	74,516	45,603	45,572	79,619	99,633
Portugal	10,937	11,335	10,614	14,064	8,373	13,196
Spain	6,187	2,979	2,249	4,099	4,063	2,478
Sweden, Norway and Denmark	22,681	24,486	24,918	28,292	20,120	18,288
Russia	22,090	15,110	17,295	15,068	15,141	16,673
United Kingdom	23,973	22,468	25,477	19,210	18,108	18,847
United States	162,505	144,817	168,919	155,996	100,000	70,332
Other Countries (*b*)	6,810	9,484	18,484	25,833	25,167	23,277
Totals	558,162	462,299	467,319	475,508	475,745	456,260
Stock in Sicily at end of year	221,204	302,410	339,113	361,220	396,541	462,437

(*a*) In 1900 and 1901 by A. S. Malcolmson, New York; for following years, by Emil Fog & Sons, Messina. (*b*) Mainly South Africa, Northern Africa, Australia and the East Indies.

The contract of the Anglo-Sicilian company expires on July 31, 1906. In order to replace the retiring company the Sicilian mine owners are going to form a new trust. As it would be impossible to unite them all by free will under one head, a law will be presented to parliament making it compulsory on all interested to form part of the new combination. This "Consorzio Obbligatorio" will dispose of the whole Sicilian production and will exercise a more complete monopoly than the Anglo-Sicilian could. Representatives of the mining interests are now treating in Rome with the Anglo-Sicilian and the Louisiana interests. They hope to obtain from the

Anglo-Sicilian an extension of time for the sale of accumulated stocks, which the Consorzio Obbligatorio is asked to take over, and to liquidate with the old company.

SHIPMENTS OF SULPHUR FROM SICILY TO THE UNITED STATES.
(In long tons.)

Port.	1900.		1901.		1902.		1903.		1904.		1905.	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.
New York....	70,446	24,307	72,104	19,631	76,383	26,842	70,800	21,201	41,429	10,547	26,782	16,270
Philadelphia..	1,600	5,100	2,300	9,595	3,500	10,399	4,910	8,500	1,325	5,825	800	1,848
Baltimore....	6,800	5,400	7,550	2,900	9,065	2,400	10,900	2,000	3,370	1,400	.....	....
Boston......	1,500	2,500	3,497	3,200	2,204	2,300	5,508	2,450	11,397	1,749	5,477	1,009
Portland, Me.	27,612	.....	21,990	.....	26,328	.....	23,855	.....	23,638	.....	17,882	.....
Other ports(*a*)	14,240	3,000	650	1,400	8,498	1,000	5,872	.....	.....	.....	.....	.....
Totals.....	122,198	40,307	108,091	36,726	125,978	42,941	121,845	34,151	81,159	19,521	50,941	19,127

(*a*) Norfolk, Mobile, New Orleans, Savannah, San Francisco, Bangor, Portland, Ore. and Canada.

WORLD'S PRODUCTION OF SULPHUR. (*a*)
(In metric tons.)

Year.	Austria. (*d*)	France. (*c*)	Hungary	Germany	Greece.	Italy. (*b*)	Japan.	Spain.	United States.
1895..................	830	4,213	102	2,061	1,480	370,766	15,557	2,231	1,676
1896..................	643	9,720	138	2,263	1,540	426,353	12,540	1,800	3,861
1897..................	530	10,723	112	2,317	358	496,658	12,013	(*b*)3,500	1,717
1898..................	496	9,818	93	1,954	135	502,351	10,339	3,100	2,770
1899..................	555	11,744	116	1,663	1,150	563,697	10,241	1,100	1,590
1900..................	862	11,551	123	1,445	891	544,119	14,439	750	4,630
1901..................	4,911	6,836	137	963	2,336	563,096	16,548	610	6,977
1902..................	3,721	8,021	105	487	1,391	510,333	18,287	450	7,565
1903..................	4,475	7,375	135	219	1,266	497,615	22,914	1,680	35,671
1904..................	6,288	5,447	143	209	569	519,243	(*e*)	......	193,492
1905..................	(*e*)	(*e*)	(*e*)	205	(*e*)	530,510	(*e*)	(*e*)	235,712

(*a*) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to between 20,000 and 30,000 long tons annually, is not included. (*b*) Crude. (*c*) Raw mineral; limestone impregnated with sulphur. (*d*) Crude rock. (*e*) Not yet reported.

The situation of the sulphur producers is such that the law referred to has every probability of being passed. The scheme is supported by the Government.

Japan.—Sulphur is widely distributed in Japan, but is most actively mined in the north; it occurs generally in the vicinity of still active volcanoes. There are six principal mines, in order of importance: Imaonobori and Tsurugkisan in Hokkaido province, Skiranesan in Tochighi, and Yahukodoyama, Doro-in and Iwojima in Sahuma. The first two belong to the firm Mitsni Buyan Kaisha, of Tokio.

The country's output has grown from 10,000 tons in 1900 to above 20,000 tons in 1904. Domestic consumption takes only a quarter of the output. Exports amount to 14,000 or 15,000 tons per year, the western coast of the United States and Australia being the largest consumers; Hakodate is the usual shipping port, although Kobe and Nagasaki ship small quantities to China. Recent quotations have been $17.50 per ton

for the best grades and $15.50 for inferior grades, with variation of 50c. per ton according to method of packing.

In northern Japan, snow interrupts activities for five months in the year. The process of extraction is still primitive; the ore is put into iron receptacles and heated to 115 deg. C. The Mitsui company, a few years ago, introduced modern Gill's furnaces such as are used in Sicily. The yield of sulphur from Japanese ore is probably the highest in the world. It reaches 50 per cent. in the north, where ore of less than 38 per cent. is rejected. In Sicily, with modern furnaces, 20 per cent. ore is considered workable.

Commercial Conditions.

The imports of sulphur of all kinds into the United States in 1905 amounted to 84,579 long tons, valued at $1,561,354, a decrease of 45,842 tons and $951,334 from the previous year. The effect of the growing home production upon imports is plainly shown in the following table:

IMPORTS OF SULPHUR INTO THE UNITED STATES.
(In tons of 2240 lb.)

Kind of Sulphur.	1903.		1904.		1905.	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
Crude, or brimstone	188,888	$3,649,756	128,885	$2,463,779	83,201	$1,522,005
Flowers	1,854	52,680	1,332	39,133	572	16,037
Refined	160	3,746	163	4,373	79	19,960
Lac, or precipitated	29	3,508	41	5,403	27	3,352
Total	190,931	$3,709,690	130,421	$2,512,688	84,579	$1,561,354

The threatened disorganization of the sulphur industry was averted by a working agreement between the Anglo-Sicilian Sulphur Company and the Union Sulphur Company which, between them, control the world's supply. The American trade is divided in an agreed ratio until August, 1906, and power is given to the Union Sulphur Company to dispose of a small proportion of its American quota in European markets. The agreement further provides for equality of selling price, based upon the sulphur content of the products sold by the two companies. This was necessitated by the fact that the Louisiana sulphur is practically refined before it reaches the surface. Sicilian sulphur, on the other hand, is mined in the ordinary way, and is then subjected to fusion, being melted out from its associated gangue. A small proportion of foreign matter, seldom exceeding 1½ per cent., attaches to the sulphur in its fused condition, and in order to place crude Sicilian sulphur upon a selling equality with refined Louisiana it is now sold upon a sulphur-content basis, by which means a parity of prices is established.

Quotations early in 1905 were $21.45 for Sicilian "seconds," and $21.75 for "prime." In March, the price of Sicilian had fallen to $21, domestic

b ing quoted at $21.25. During April, freights were quoted as follows: From Louisiana to Europe, $3 per ton; from Louisiana to Maine, $2.75; from Sicily to New York, $1.75 per ton. During April, domestic readjusted itself at Atlantic ports as follows: New York, $21 per ton; Baltimore and Philadelphia, $21.50; Portland, $21.75. These prices, with slight fluctuations, held about the same until the middle of the year; when domestic settled to $20.50 per ton for the ports of New York, Boston and Portland, and $20.75 for Philadelphia and Baltimore. During the later months Sicilian prices were the same as those for the domestic product. The price of sulphur advanced from the middle of the year, by 50-cent steps, and at intervals of a few weeks, to the closing figures—$22.25 for New York, Boston and Portland; and $22.75 for Philadelphia and Baltimore.

Pyrite.

The two great industries that consume sulphur and pyrite are the manufacture of sulphuric acid, and the preparation of sulphite liquor for digesting and bleaching wood pulp. In this country, until recently, sulphur has been used exclusively in the latter industry, although pyrite is claimed to have equal, or superior, advantages. Arguments on both sides of this question have been presented by Herman Frasch, President of the Union Sulphur Company, favoring the general use of brimstone[1], and by Ernst A. Sjöstedt, metallurgist for the Lake Superior Power Company, who records[2] in particular the successful and money-saving employment of pyrite at this company's plant at Sault Ste. Marie.

PRODUCTION, IMPORTS AND CONSUMPTION OF PYRITE IN THE UNITED STATES. (*a*)
(In tons of 2240 lb.)

Year.	Production.		Imports (*b*)		Consumption.	
1896	109,282	$292,626	199,678	$140,571	308,960	$1,433,197
1897	133,368	404,699	259,546	847,419	392,914	1,252,118
1898	191,160	589,329	171,879	544,165	363,039	1,133,494
1899	178,408	583,323	310,008	1,074,855	488,416	1,658,178
1900	201,317	684,478	322,484	1,055,121	523,801	1,739,599
1901	234,825	1,024,449	403,706	1,415,149	638,531	2,439,598
1902	228,198	971,796	440,363	1,650,852	668,561	1,622,648
1903	199,387	787,579	425,989	1,628,600	625,376	2,416,179
1904	173,221	669,124	413,585	1,533,564	586,806	2,202,688
1905	200,280	651,796	515,722	1,780,800	716,002	2,432,596

(*a*) These statistics do not include the auriferous pyrite used for the manufacture of sulphuric acid in Colorado. (*b*) Net imports, less re-exports of 3060 tons, in 1902, and 1330 tons, in 1903.

Market.—The market price of pyrite was regular during 1905, domestic 38 to 45 per cent. lump ore ranging from 9.5 to 10.5c. per unit of sulphur ($3.99 to $4.41 per long ton), and fines, 8.5 to 9.5c. per unit ($3.57 to $3.99 per ton); while imported 42 to 56 per cent. lump ore was quoted at 10 to 12.5c. per unit ($4.90 to $6.13 per ton), and fines, 9 to 11c. per unit ($4.41 to $5.39 per ton), all f. o. b. Atlantic ports.

[1] *Engineering and Mining Journal*, Dec. 23, 1905.
[2] *Ibid.*, April 28 1906.

Pyrite Mining and Milling in Virginia.

By Robert K. Painter.

Pyrite carrying more or less copper is found along a comparatively narrow belt, extending from a point about 30 miles south of Washington, thence about S. 30 deg. W. throughout the middle and south-Atlantic States; the deposits have been largely developed only in Virginia. The mines of note are near Mineral, Louisa county, and Dumfries, Prince William county.

The deposits consist of a series of lens-shaped bodies lying nearly conformable with the enclosing country-rock of slate and schist. The lenses are of varying size; in the Louisa mines, lenses several hundred feet long and, at points, as thick as 80 ft., have been worked; in the Cabin Branch mine, near Dumfries, 10 ft. may be taken as the maximum thickness, although one lens has attained a length and breadth of nearly 1000 feet.

The ore is sold on a basis of the sulphur content, and is prepared for market in three sizes, lump, spall and fine. Lump is the first-class ore as broken in the mine, but free from adhering slate or lean material. Spall is clean lump broken to burner size (passing a 2½-in. ring) from which the fine and smaller size, produced by breaking, have been screened. Fine is ore which will pass a screen having an aperture of about ⅜ in. It is usually washed or jigged to bring it up to shipping grade.

The general practice is to develop the orebodies by inclined shafts. In the Louisa county mines, stopes are opened up by driving levels along the lens at economic intervals, which are then connected by raises and the ore broken down by overhand stoping. As the walls are comparatively strong, little timber is used, and the levels are protected by massive pillars with small openings into the stopes, through which the ore is drawn to be loaded into mine cars. Haulage is by mules. The mine cars are dumped into the skips. Machine drills are used largely, and little sorting is done underground.

In the Cabin Branch mine, the conditions have caused somewhat different methods to be adopted. The walls are very soft, the ore often much faulted, and the dip variable. The mine is opened by an inclined shaft following the ore in a general way, but often cutting into the slate to some extent to avoid the sharper bends which the orebody makes. Levels are run off at intervals, varying from 40 to 90 ft. according to the dip. The driving of these to the extreme end of the lens is pushed as fast as possible, an occasional raise being made for ventilation, which is widened out into a flask-shaped stope as drifting proceeds. As the stopes are lengthened, "breakthroughs" are made into the levels, every 30 ft., from which the mine cars are loaded. When the dip is steep enough to allow the ore to slide, chutes are employed. As the ore is extracted, the roof is held by round pine props or stulls. A thin rib of ore is left at the top of a

stope to carry the track above, until that level is stoped out. Draw slate is packed away in the middle of the stope, and held by cribs when necessary. When a level has been thus stoped out, robbing of ribs and stumps begins at the outer end, retreating toward the shaft. When the props hold up the back over too great an area, they are shot out to keep the ground caved nearly even with the retreating party. By this method, little ore, except that shot into the slate packs, is lost in mining. As much slate as possible is picked out and left in the stopes. Machine drills of 2½-in. size are used for development, and for stoping, where the ore is thick enough. Where the ore is thin, and in robbing, hand drills are used to better advantage. Some 2-in. drills, run by one man, are used in stoping with fair success. As the dip is seldom over 40 deg., no level pockets are provided at the shaft, but the mine wagons dump direct into the skip, which is of 3000-lb. capacity. Tramming is done by hand.

The larger part of the output of the Louisa mines consists of fines. One mine turns all its ore into fines. At this mine, the skip dumps upon a grizzly of railroad iron set 3 in. apart. The slate is picked and forked from the two sizes, which go to their respective crushers at the head of the mill. By successive crushing with breakers and rolls, and screening by trommels, the ore is here reduced to jig size and fed by a distributor to the Harz jigs. Jigging through a bed of cast-iron balls is employed, and the hutches discharge upon a belt-conveyor of ascending grade, which delivers to the stock piles, where the ore soon drains.

At another mill, the skip dumps upon a similar grizzly; the oversize is sorted into No. 1 lump for shipment, No. 2 lump to the mill, slate to waste; the undersize falls upon a floor, where it is forked and the slate picked out, the remainder going to the mill in a separate car by rope haulage. Some of the lump, generally the better grade, goes to a jaw-breaker set to make spall ore, a fine-bar grizzly cutting out that under 1¼-in. size, which goes to the mill. The spall ore is loaded in box cars, after being inspected and having the slate thrown out. The mill is arranged to crush the ore by successive steps, in Blake crushers, roll-jaw crushers and rolls, with repeated screening, to jig size. The same practice (of jigging through a bed of cast-iron balls) is employed; but the jigs discharge into a dewatering device which feeds a belt-conveyor delivering to the stock pile, bins or a hearth drier, depending upon the weather. Tables are being installed to treat the overflow from the dewatering tank.

At the Cabin Branch mine, the skip dumps upon a 2½-in. grizzly. The oversize goes to the lump-ore storage-bin, from which it is drawn off at intervals and sorted. Slate goes to the dump, first-class lump to the spalling floor and the remainder to a roll-jaw crusher at the head of the fine-jigging annex. The No. 1 lump is all spalled by hand (to avoid excess production of fine) to burner size, and forked into measuring chutes delivering

to the cars. This spalling is done by contract, the men using long-handled hammers of 2-℔. weight, and working upon a thin layer of ore upon the floor at one time. The resulting fine is small in amount and is screened into the fine bin. The undersize from the grizzly goes to a revolving screen with 1¼-in. round holes, through which the ore is washed into the pebble-ore jig. The oversize goes to a slate-picking chute delivering upon the spalling floor. The pebble-ore jig is a three-compartment Harz jig fitted with No. 2 mesh, No. 8 iron-wire screens. It discharges, from the spouts, clean pebble and middling; from the hutch, clean fine, middling and waste. It handles up to 75 tons per day at 150 r.p.m. The clean pebble is elevated to a screen delivering to a shipping chute, the small ore being returned to the roughing rolls. The clean fine runs into the fine bin, which is heated by exhaust steam. The fine middling is elevated to the roughing rolls. The crushed ore, after passing through rough rolls, meets the No. 2 lump which has passed the roll-jaw breaker, and the product from the fine-crushing centrifugal rolls, and is elevated to a two-unit centripact screen. The upper screen is 2⅜-in. aperture, and the oversize goes to the roughing rolls; the second screen is ⅛-in., and the oversize goes to the fine rolls; the through size goes to the fine jigs (which are of two compartments), jig through a bed of iron balls, and discharge into the fine bin. The installation of spiral slate-pickers (to rough-pick the 1½-in. pebble-ore) is being considered.

The total cost per ton of clean ore (f. o. b. cars or vessels at shipping point) depends upon the distance of the mine, and the size and quality of the deposit, but average costs based upon the "ton of mine-dirt" hoisted, cleaned at the mine, and loaded ready for transportation to the final shipping point will be about as follows.

For a deposit where the lenses measure some hundreds of feet in length and breadth, with an average thickness of 5 ft., the table given herewith may be taken as representing good Southern practice for a mine hoisting about 4000 tons per month from moderate depths.

COST OF MINING, PER TON HOISTED.

Labor	$0.77
Timber	.01
Powder, etc.	.12
Drill parts, pipe	.01
Fuel	.06
Oil and waste	.03
Tools and supplies	.03
	$1.03

DISTRIBUTION OF MINING COSTS.

Constant (power, foremen, engineers, etc.)	$0.25
Mining (stoping, tramming, timbering, etc.)	.49
Development (sinking, drifting, etc.)	.29
	$1.03

COST OF ORE-DRESSING, PER TON HOISTED.

Labor	$0.41
Fuel	.04
Supplies	.02
	$0.47

GENERAL EXPENSE, PER TON HOISTED.

Superintendence, supply house clerk, watchman, etc.	$.10

SUMMARY.

Mining	$1.03
Dressing	.47
General expense	.10
	$1.60

These figures will be increased by the cost of transportation to, and loading upon, cars or vessels; taxes; depreciation; selling ore, etc. The total

will be increased, moreover, in direct proportion to the amount of waste in the mine-dirt hoisted. Thus ore from which only 50 per cent. is dressed would cost for the above operations $3.28. It will be seen, then, that unless a mine be situated favorably with regard to shipping facilities or contains orebodies of large size, it should produce ore running not less than 50 to 60 per cent. pyrite to be attractive.

Pyrites Mining and Milling in St. Lawrence County, New York.

By Robert B. Brinsmade.

The pyrites deposits extend from the Cole mine, four miles north of Gouverneur, to the High Falls mine, 14 miles northeast. The orebodies vary locally, but in general are bedded veins conformable to the schist, containing pyrite in a quartz and feldspar gangue. Pyrrhotite occurs in quantity in the High Falls mine.

The three producing mines are the Cole, the Stella and the High Falls. The Cole mine, owned by the Adirondack Pyrite Company, of Gouverneur, has been opened within a few years and has been producing more or less steadily. So far the development has shown a body 100 ft. long by 60 ft. wide, extending overhead to the surface and to unknown depth. The hanging wall is good, and there is no need for roof support by timber. The formation permits a machine-drill to do 70 to 100 ft. of holes in 10 hours. These conditions contribute to a cheap extraction of the ore.

The Cole concentrator is at present the best equipped on the range. The mine skip dumps directly on a platform before two 14x18 in. Blake crushers (one is a relay), from which the ore is spouted to a set of 12x24 in. Cornish rolls, with steel-spring buffers. Water is fed to both crusher and rolls. From the rolls the material is raised by a 14-in. rubber-belt bucket-elevator to a trommel, 36 in. diameter and 6 ft. length, with $\frac{3}{16}$-in. round holes; this is set at 1:20 slope, and is connected to its driving pulley by a flexible joint. The trommel oversize is recrushed in 14x19-in. rolls, and fed back to the trommel head by a second bucket elevator.

The trommel undersize, $\frac{3}{16}$-in. particles and less, descends infour equal streams to four Harz jigs, alike in form and adjustment. Each jig has three (24x32 in.) beds, with four-mesh No. 16 wire screens. Each bed is 4 in. deep at the lower and 5½ in. at the upper end, below the parting weirs. The jig pistons are wood, with rubber-belt packing rings, and are actuated by the usual eccentrics. The concentrate is drawn both off the screen and from the hutch into a central flume leading to a bucket elevator. The elevator raises the concentrate to the top of the mill and spouts it either directly through a 3-in. pipe to load box-cars direct, or to elevated storage bins. The jig tailing runs to the boot of a centrifugal sand pump, which elevates it to a flume leading to the tailing heap outside.

The raw ore is usually 20 to 30 per cent. sulphur, and the concentrate goes

40 to 50 per cent. The tailing assays over 5 per cent. in sulphur. Trials at the Stella mine on similar ore showed that graded sizing and hydraulic classifying before jigging did not make appreciably higher-grade concentrate than the system above described. The tailing was lower, but because of the added attendance, interest and repairs, due to increased apparatus, the profit was less. Also, there was more slime saved, but it is not desired by pyrite burners. In spite of this last disadvantage, it ought to pay to make leaner tailings. This could be done by working the last of the three jig beds thin to produce a middling product. These middlings (half pyrite and half gangue, that now go to waste) could then be reground in rolls and retreated on fine jigs or Overstrom tables.

The following is the labor cost of operating the Cole mine and mill for a production on an average run of 55 tons concentrate from an average of 110 tons of raw ore:

Mining—Two shifts of 10 hours each.	
1 pit boss	$2.50
4 Italian muckers	6.00
2 American muckers	3.50
One shift	$12.00
Two shifts	$24.00
2 rock drills, day shift, on contract @ 7c. per ft.—140 ft. average	9.80
Total mine labor	$33.80
Per ton raw ore	.307
Per ton concentrate	.614

Milling—Two shifts of 10½ hours net, each.	
1 engineer	$2.92
1 fireman	1.75
1 shift boss	2.75
2 jigmen	5.50
1 head crusherman	1.90
2 assistant cruchermen	3.30
1 car loader	1.90
One shift	$20.02
Two shifts	$40.04
Per ton raw ore	.37
Per ton concentrate	.74

The labor cost for mining is reasonably low, but for milling it is excessively high. In southeast Missouri, with more elaborate apparatus and a closer saving, the total cost per ton of raw ore need not exceed 30c., as compared with 33 per cent. more here for labor cost alone, due largely to the inexperience of the mill men.

The Stella mine, which was the first opened of the pyrite mines of this district, was actively worked until the failure of its owners in 1900. It remained idle until after the burning of the concentrator in 1904, when the property was sold to the St. Lawrence Pyrite Company. There are five known parallel veins on the property, with a northeast strike, but only two have so far been developed. To date, all shipments have been from the western vein, whose incline adjoins the site of the destroyed mill. This incline is 16 ft. wide, and 700 ft. deep, at an angle of about 30 deg. It is devoid of timber, the mica schist hanging wall being strong enough to permit stopes 50 to 80 ft. wide between pillars. The stopes were excavated by driving drifts every 30 to 50 ft. along the incline and removing the ore by breast stoping and raising around the rock pillars necessary to support the roof.

The St. Lawrence Pyrite Company is now preparing plans for a new concentrator and through a subsidiary company will renovate the old branch railroad extending 2½ miles to the main line of the R. W. & O. railroad at

DeKalb Junction. When the present plans are carried out the Stella will be the largest producer on this range.

The High Falls mine has recently been transferred to the National Pyrite Company of Canton. The concentrator merits little technical study, inasmuch as it has only two jigs, arranged with crusher and trommel in a similar, but less effective, way than in the Cole mill. The capacity is 50 tons of raw ore, from which 20 to 30 tons of concentrate is shipped.

The future of this district depends chiefly on commercial considerations. There is plenty of pyrite ore to be mined. A considerable market exists in the sulphite pulp industry of the Adirondacks. Many shipments have also gone to sulphuric-acid works. The pyrite contains no impurities objectionable to acid makers.

TALC AND SOAPSTONE.

Talc and soapstone are marketed in various forms, the most important of which are rough blocks, slabs (sawed), manufactured articles, and pulp (powder). The manufactured articles include bath and laundry tubs, brick for stoves, heaters, etc., hearthstones, mantels, sinks, griddles, slate pencils, tailors' pencils, gas tips, and numerous other articles of common use. The ground talc is employed for paper making, for foundry facings, as a lubricant, for dressing skins and leather, etc. By far the largest portion of the production of talc in the United States is the fibrous mineral which is mined in New York. This is used chiefly for paper filling, in connection with the wood pulp industry; it is used to a less extent for the manufacture of wall plaster, and in the preparation of paint.

Outside of New York talc is produced in New Jersey, Pennsylvania, North Carolina, Virginia, Massachusetts, Vermont, California, Washington and Georgia. The production of talc in Pennsylvania and New Jersey, North Carolina and Virginia is of considerable importance; that of the other States is insignificant. Virginia is the chief source of the talc, or soapstone, that is sawed into slabs.

The production of fibrous talc in New York in 1905, according to the statistics of the State Geological Survey, was 67,000 short tons, valued at $469,000, an average of $7 per ton, against an average of $7.93 per ton in 1904, and $7 per ton in 1903. The occurrence and technology of talc in New York is described in a subsequent paper. The general statistics of the talc and soapstone industry are given in the following table:

STATISTICS OF TALC AND SOAPSTONE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.						Imports.		
	Fibrous Talc.			Talc and Soapstone.			Sh. Tons.	Value.	Value Per Ton.
	Sh. Tons.	Value.	Per Ton.	Sh. Tons.	Value.	Per Ton.			
1896	51,816	$256,080	$4.94	21,448	$207,085	$9.66	1,950	$18,693	$9.60
1897	52,836	283,685	5.37	27,068	259,948	9.60	779	8,423	10.54
1898	54,807	285,759	5.21	27,974	237,280	8.48	445	5,526	10.70
1899	57,120	272,595	4.77	26,682	241,267	9.04	254	3,534	13.91
1900	45,000	236,250	5.25	26,726	249,777	9.35	79	1,070	13.50
1901	69,200	483,600	6.99	28,643	424,888	14.83	2,386	27,015	11.32
1902	71,100	615,350	8.65	26,854	525,157	19.56	2,859	35,336	12.36
1903	60,230	421,600	7.00	26,671	418,460	15.69	1,791	19,677	10.99
1904	64,005	507,400	7.93	27.184	433.331	15.94	3,268	36,370	11.13
1905	67,000	469,000	7.00	40,134	637,062	15.87	4,000	48,225	12.06

(a) Statistics for 1902 and subsequent years, except 1905, are as reported by the United States Geological Survey.
b) The value of these products has not much significance owing to the diverse conditions of the materials reported.

Talc in Northern New York.

By Robert B. Brinsmade.

About six miles east of the pyrite range of St. Lawrence county extend some low ridges of upturned metamorphic strata which contain a remarkable series of talc deposits. On the surface the talc veins are discolored, but further down is disclosed clear greenish-white talc, free from injurious impurities.

The talc range proper has a length of seven miles and a width of over a mile; the strike is northeast through the towns of Fowler and Edwards. On the western end of the range, two miles south of the village of Little York, are the American and Columbia mines; these belong to the Union Talc Company, of Gouverneur; from three mills there is an output of over 30,000 tons of ground talc a year. In the center of the range are the three mines of the Ontario Talc Company, with one mill, whose capacity is 20 tons daily. At the eastern end, at Talcville, there is a workable vein a mile long. The United States Talc Company owns the western part of the Talcville vein, with a mill at Dodgeville, of 15,000 tons yearly capacity. The remainder of the Talcville vein belongs to the International Talc Company, with four mills, only three of which are operated.

The talc bed that forms the working mines is 12 to 16 ft. thick in the Union Talc Company's mines, and dips 30 to 50 deg. to the northwest. The vein here is white agalite, foliated or massive; these varieties are ground in varying proportions for different purposes. The foot-wall is limestone; the hanging wall a thin stratum of a biotite hydrous mica-schist.

At Talcville the vein dips 45 to 60 deg., and has two layers separated in places by a slip; the average width is 18 ft., but occasionally it is workable for a thickness of 70 ft. Here, too, the talc, while equally hydrated, retains physically much more of the fibrous structure of its parent tremolite than at Little York; hence it is more valuable. The scaly and lamellar rensselaerite (of a beautiful greenish-white tint, and which renders the ground talc too ''leaf-like'' for the better grades of paper) occurs chiefly at the west end of the district.

Mining Practice.—The American mine furnishes a good example of the prevalent extraction method. It is opened by an incline, following the foot-wall of the vein to a depth of 300 ft.; for its 16-ft. width it stands well without timbering. The hight of the incline is the thickness of the talc, or 15 ft. In this incline is a single skip track and a stairway for men; every 30 to 50 ft. on each side are opened wide arched tunnels 15 ft. high. After the tunnels have advanced about 25 ft. from the side of the incline, an upraise is started to the next level; when the raise is ''holed through,'' the talc is stoped out clean between walls, by underhand stoping, for a distance of 40 ft. along the vein. The stope is then squared down to the lower

level, and tunneling is begun under the next pillar, preparatory to another upraise and underhand stope beyond. It is not usual to extend the tunnel and its accompanying stopes more than 600 ft. each way from the incline, as it is claimed that the tramming expense then becomes too heavy for economy.

Usually the bottom level is the only one in operation; but by leaving a shelf of talc on the foot-wall of the upper level to support the car track when the raise comes up from the lower level, as many levels as may be wished can be kept open with this system, if the face of the upper level is always kept in advance of the one below. The pillars are left 25 to 30 ft. square, as the cleavage of the talc makes round pillars unsafe. By leaving a foot's thickness of talc under the mica-schist layer on the hanging wall the roof stands well, so that no stulls nor special "roofmen" are necessary.

The capacity of the American mill is 32 tons; for this it takes a mine force of 1 shift boss, 1 drill runner, 1 helper and 4 muckers, working the day shift only. The drill used is a No. 2 Rand, and it works on one side of the incline while the muckers are cleaning out the broken rock (from the previous day's blasting) on the other. The talc is regular and easy drilling ground; one shift can break 60 to 70 tons. This is greatly facilitated by its cleavage, along bedding planes, that causes the mineral to split off in huge slabs. A hole is loaded with 40 per cent. dynamite; this is tamped by a clay roll, 12 in. long, sent down from the surface, and is fired with cap and fuse. Care is taken that only clean talc shall be broken down; this is usually an easy matter for the whole width of the vein, except for occasional tremolitic "horses."

The broken talc is run out, in one-ton wooden cars, to the incline, and is dumped direct into a one-ton steel skip. This is pulled up to the surface by a ¾-in. steel rope. There, the mineral is dumped automatically over a car running on a trestle through the storage sheds. These sheds hold several hundred tons reserve. Here the different kinds of talc can be mixed for the mill, and any chunks containing quartz or other foreign and deleterious matter can be rejected before they enter the mill to injure the grinding machinery.

At the Talcville mines, the practice does not differ materially from that just described. In the very wide stopes that sometimes occur, it is often necessary to run two parallel tunnels under the pillars (instead of one); this plan leaves an enclosed talc wall that will support a back too wide for one continuous arch.

The Oswegatchie river flows through the talc region; its steep gradient, accompanied by numerous rapids and several falls, permits the easy utilization of its water power. The river here has a flow of 500 to 1000 cu. ft. per sec., giving about 50 h.p. for each foot of water head on the turbines. With the exception of two that run by steam, the talc mills are located on

this river at convenient water-power sites; the ore comes to the mills from the Talcville mines on the Gouverneur & Oswegatchie railroad. The mines off the railroad haul the talc to mill usually with two-horse teams.

Milling Practice.—The American mill is arranged in three floors like a flour mill. The broken talc from the storage sheds of the American mine, about 150 yd. distant, is dumped from a 1-ton car on a trestle in front of a 10x20-in. Blake crusher, run at 240 r.p.m. The crusher spouts directly into an 18-in. cone or "coffee-mill grinder," driven 640 r.p.m. from a bevel gear at the top of its vertical shaft, the central cone being cast in two halves to cheapen renewals. From the cone, the talc pieces, now about 1 in. diam., are raised by a 6-in. bucket-elevator (having, like all elevators in these mills, a canvas belt) to the boot of a 10-in. bucket-elevator discharging into the storage bins, between the second and third floors, that feed the Griffin mills. The Blake and cone crushers are placed between the first and second floors. This crushing plant has capacity enough in 10 hours' work to keep the balance of the mill supplied for 24 hours.

On the first floor are three Griffin mills (one being used only as a relay) with a capacity of three tons hourly apiece to 60-mesh, when driven 180 r.p.m. by a 14-in. leather belt. The mill discharge is sucked through a 12-in. round galvanized pipe into the top of one side of a steel settling hopper (of 10 in. diam., with a conical bottom of a slope of 45 degrees) by a steel centrifugal fan placed at the opposite side. The coarser talc settles in the hopper cone, while the fine dust is drawn across through the fan and blown into the settling chamber. This chamber occupies one-half of the third floor, being separated from the rest by a light canvas partition to permit the exit of air but not of dust; the latter is fine enough for shipment and is swept off the chamber floor once a day.

The talc in the settling hopper is taken by a screw conveyor to an adjoining trommel. This trommel is octagonal, about 3 ft. diam. and with a fall of 6 in. in its length of 16 ft.; it revolves 30 r.p.m. and is covered with 26-mesh wire-cloth. The trommel oversize is rejected; but the undersize descends in flexible spouts to small cars (on the second floor) that run on tracks just over the six Alsing cylinders.

The Alsing cylinders are 6 ft. diam., 8 ft. long, and rest on horizontal journals which revolve 20 r.p.m. Each cylinder is charged with 2000 ℔. of talc and 6000 ℔. of pebbles; it revolves 4½ hours. A small central door in the cylinder is then removed and a grating (to retain the pebbles) is put in its place. The cylinder is given 15 min. more turning, when the talc dust is all emptied into a wooden hopper above the first floor. It thus takes six cylinders to grind 12,000 ℔. in 4½ hours, giving a plant capacity of 32 short tons in the 24 hours. The cylinder lining is of porcelain tile; this is 2½ in. thick, is made to dovetail together, and should last about five years. The pebbles are stream flints, about 2 in. diam., shipped from

Labrador or Greenland; they cost $21 per ton. The heat generated in grinding is considerable, but the only harm done is the occasional splitting of the pebbles, that are consumed to the extent of 1000 lb. annually per cylinder.

The ground talc is now "spouted" from the Alsing hoppers to small cars, which in turn dump to the boot of an 8-in. bucket-elevator; this discharges into a second trommel on the third floor. This trommel is cylindrical, and of the same size and speed as the one already described, but with a finer wire-screen. The oversize (mostly wood fiber or broken pebble) is rejected; the undersize descends to bagging bins above the first floor. From these last bins the finished talc is drawn into 50-lb. paper bags for market.

On each 12-hour shift in the mill are one engineer, one machinery oiler, and two baggers; while on the day shift, in addition, there are a foreman. a crusher car-loader, and a crusher feeder; or a total of 11 men in the 24 hours.

The United States mill at Dodgeville, one of the newest plants, differs in several important respects from the American mill. Located just below a dam on the river, the power is derived from a canal and horizontal shaft turbines. The mill building itself is 150 ft. long by 80 ft. wide, and has three floors, only two of which occupy the full ground area of the structure. Here the discharge of the Blake crusher is fed to 30-in. Cornish rolls (instead of to a cone mill); the rolled material is elevated to four buhr-stones (instead of to Griffin mills). The buhr-stones are in four pairs, each stone being 3 ft. in diam. by 6 in. thick, and constructed by Munson & Co., of Utica. A conveyor on an incline of about 10 deg. (with 2 in. square wooden scrapers placed 22 in. apart on a canvas belt running in a steel trough) now raises the ground talc on to a similar horizontal conveyor; this distributes the talc over a long storage bin feeding the Alsing cylinders. These scraper conveyors do the work efficiently; they obviate the need for a large settling chamber and the extra power required by the suction-fan system for removing the grinding-mill discharge.

The ten Alsing cylinders here are of the same size as those at the American mill; but an innovation is the screw-conveyor delivering the discharge from the wooden hopper underneath each pair of cylinders to the boot of a bucket elevator that lifts the talc dust to the final cleaning trommel, above the bagging bins. Other improvements are the automatic scales on the bagging bin spouts, and an inclined bag conveyor for lifting the 50 lb. sacks of talc from the storage room to the shipping platform.

The paper mills are the chief consumers of the ground talc; the talc is very efficient for paper filling, not only on account of its fibrous felting quality, which strengthens the paper, but also because talc can be assimilated by paper stock to double the weight of its non-fibrous competitors.

Less important demands come from the makers of cheap soap, of waterproof paints, of non-conducting pipe-covering and plaster, of face and foot-powders, and of adulterated drugs.

The value of the talc depends upon the color and the fibrous quality, so that physical and not chemical tests are used for grading. The better grades have a pure white with a bluish tinge; the poorer have a yellowish cast. To the naked eye the shipping product of under 100-mesh fineness appears like an impalpable powder; but under the microscope the varying fibrosity of the particles is plainly seen.

The productive power is limited solely by the grinding capacity of the mills, as the mines can respond readily to any demand likely to be put upon them.

TIN.

BY W. R. INGALLS.

THE price for tin ruled high throughout 1905, and during the week ending April 11, 1906, it attained the highest figure on record, the quotation having been 38.6c. per lb. In January and February, 1888, the price of 37c. was reached, which up to that time was the highest on record, so far as I can determine. The price statistics for the early part of the last century are by no means complete.

The rise in tin has been fundamentally due to a strong statistical position; in other words, an exhaustion of accumulated stocks and a demand which can be satisfied by supply only at enhanced prices.

The world's production of tin in 1905 showed a decrease of about 3000 tons, which occurred chiefly in the outputs of Banca and the Straits Settlements. In the latter country, which is the source of about 60 per cent. of the present supply of tin, the outlook is not at all promising, the cost of production having increased both on account of the exhaustion of the richer and more easily worked alluvial deposits, and also by the rise in the value of silver, and disturbances in connection with the currency question in general. It is believed that production has reached its maximum, and unless new discoveries of importance are made, there is a probability of further decrease in the output during 1906.

The situation naturally raises the question where is the world's requirement for tin going to come from? At the present time, Bolivia is the only country which is showing any material increase in production, and its increase in 1905 was not very large. However, production there was hampered by drought, while the lack of cheap transport facilities is always a difficulty. The former was only a temporary drawback, and the latter will be remedied. The Bolivians are anxious to become the largest tin producers of the world, and it is not impossible that eventually they may attain that position, but it will not be in the near future.

The high price for tin is naturally stimulating developments in all parts of the world, especially in Tasmania, Victoria and other Australian States, where there are large deposits of exceedingly low-grade ore.

Moreover, it is reported that certain ancient mines in Saxony, which have been idle for many years, are now being reopened. Renewed attention is being given to the known occurrences of tin in the United States.

The production of tin ore in the United States in 1905 was insignificant. In 1904 it amounted to 159 tons, the output being entirely shipped

to Europe for smelting. The smeltery of the International Tin Company, at Bayonne, N. J., continued idle, for lack of ore supply.

IMPORTS OF TIN INTO THE UNITED STATES.

Year	Pounds.	Value.	Year.	Pounds.	Value.	Year	Pounds.	Value.
1897.......	55,172,571	$7,415,933	1900....	69,989,502	$19,458,586	1903....	83,133,847	$22,265,367
1898.......	62,748,399	8,770,221	1901....	74,560,487	19,024,761	1904...	83,168,657	22,356,896
1899.......	71,248,407	16,748,107	1902....	85,043,353	21,263,337	1905....	89,227,698	26,316,023

A good deal of tin is recovered by stripping tin-plate scrap, the alkali stannate method and electrolytic precipitation being generally employed. According to Mennicke[1], about 1000 tons of scrap are worked annually in Switzerland, 2500 to 3000 tons in Italy, and 40,000 tons in Germany. England alone produces 20,000 tons of scrap. Tin-plate scrap carries 1.5 to 3.5 per cent. of tin; the average is probably about 2 per cent. Statistics for the United States are unavailable. Detinning is carried on here by Johnston & Jennings Company, of Cleveland and Chicago; Ammonia Company, of Philadelphia; American Can Company, of New York; Continental Can Company, of Syracuse; Vulcan Detinning Company, of Sewaren, N. J.; and by several small concerns at St. Louis, and elsewhere.

Tin Mining in the United States.

Tin is known to exist in several parts of the United States, and at various times efforts have been made to exploit the deposits. This has been done in Virginia, the Carolinas, South Dakota, California and Alaska. So far, however, all the attempts have been failures, some of them disastrous failures, involving the loss of much money. It has come to be the general opinion, therefore, that the United States has no important tin resources, and never will be a producer of tin. Nevertheless, there are certainly some tin prospects in the United States which are promising. Indeed, there are several which may reasonably be thus classified, and there are at least two properties which it is believed can be profitably worked. One of these is the Harney Peak mines, which are unfortunately tied up by litigation.

Tin ore is usually of low grade. There are a few lode mines in various parts of the world which may be considered rich, as for example the Mount Bischoff (of Tasmania), which has paid large dividends from ore yielding one per cent. tin, but many mines are worked successfully on a much lower average yield.

The indications point to the Black Hills of South Dakota as the most promising region of the United States for tin production. There is no question whatever that tin exists there in considerable quantity, and over a wide area. Some of the mineralizations are also extensive in size. The mineralization is, however, irregular; and few of the mines have been

[1] *Zeit. f. Elektrochemie*, March 30, 1906

developed sufficiently to prove the probable yield of the ore in actual mining. The mineralization is such that determination of the probable grade of the ore by mine sampling is almost impossible. The conditions are somewhat analogous to those which exist in the copper mines of Lake Superior and the lead mines of southeastern Missouri, where the actual grade of the ore can be reliably determined only by actual mining and milling. The same condition appears to exist at the tin prospects at Irish Creek, Rockbridge county, Virginia.

It is probable that ore yielding 10 lb. of metallic tin per ton can be produced in South Dakota. This would indicate a value of $3.60 per ton of ore at the present high price for the metal, and a value of $2.38 per ton at the average price for the last 10 years. If tin should fall again to the level reached during the nineties (the average price at New York in 1896 having been 13.29c. per lb.) the exploitation of a mine yielding only 10 lb. per ton would be doubtful, but in view of the present situation in tin production, a high level of prices is reasonably to be expected. This is leading to the exploitation of low-grade tin ore in various parts of the world, especially in Tasmania, where careful management enables the Anchor mine to be operated successfully on a yield of only 3.5 lb. of tin per ton. The success of some of these enterprises, together with the success in treating the low-grade ores of other metals, should direct attention to the low-grade tin ore of South Dakota.

The concentration of the low-grade tin ore of the Black Hills formerly gave much trouble because of the association of the cassiterite with the mica of the country rock, the latter being a coarse greisen. It is probable that modern methods of ore concentration would succeed in overcoming this difficulty. It is to be hoped that some effort will be made to test thoroughly the possibility of tin production in the Black Hills.

The production of tin ore in the United States in 1905 continued to be insignificant, although several operations were in progress. The United States Tin Mining Company, which now has possession of the Temescal mine, near Riverside, southern California, reworked some of the old tailings, and tried to smelt the concentrate, but did not succeed in this operation.

In South Dakota, the property of the Tinton Tin Company, at Tinton, was examined by engineers, but this did not lead to resumption of operations, although it is believed that this property contains large bodies of ore which will yield about 10 lb. of metallic tin per ton. The Gertie mine was purchased by a Philadelphia company, which erected a small experimental mill. However, there was no production of tin ore in South Dakota in 1905.

Some developments, and a small production, were made in the Carolinas.

According to L. G. Craton,[1] the Carolina tin belt, as at present explored, extends from Lincoln county, N. C., to Cherokee county, S. C., a distance of about 35 miles. The belt is co-extensive with the occurrence of dikes of pegmatite, and varies from a quarter of a mile to two miles in width. The pegmatite dikes are very irregular in direction, width and extent. The cassiterite occurs in certain of them, as crystalline grains and masses, and is an original constituent of the rock. This mineral is not evenly distributed throughout those dikes in which it occurs, but rather is found in shoots.

Three mines have been opened on deposits of this kind, viz., the Ross mine, at Gaffney, S. C.; the Jones mine, near Bessemer City, N. C.; and the Faries mine, at Kings Mountain, N. C. The Jones mine is nearly 200 ft. deep.

Development has not yet been sufficient to enable reliable conclusions to be drawn regarding these deposits. The belt, however, appears to offer a promising field for exploration. Some of the developments seem certainly to have demonstrated the existence of orebodies of considerable extent, which ought to prove valuable if systematically and wisely developed. Owing, however, to the irregularity and uncertainty of extent of the deposits, cautious methods must be followed.

Alaska tin ore continued to be a very uncertain factor. The Bartels Tin Mining Company, at Cape Prince of Wales, did some development work. About 20 men, who are paid $5 per day and more, were employed. A 10-stamp mill was erected by the company at Tin City, on Behring Sea, 138 miles northwest of Nome, but it was completed too late in the year for any report to reach the New York office of the company as to its operation. Some work also was done during 1905 by Crim, Randt & O'Brien at Lost River.

At the May meeting of the New York section of the Society of Chemical Industry, Martin Schwilter spoke briefly of the discovery of cassiterite on the Seward peninsula. The tin discovery is within 30 miles of Cape Prince of Wales, which is the most westerly point of the peninsula, and about 160 miles northwest of Nome. The inland country is not well explored, but consists of mountains 2000 or 3000 ft. high, rising abruptly from the sea and being rather destitute of timber. The summers are moderate, but the winters are cold and stormy for several months; although the climate is severe, yet it is not as bad as popularly supposed. Supplies are shut off from November to June. During the open season, steamers ply between Nome and Seattle, and between Nome and San Francisco. In 1901, W. C. Bartels (of the Bartels Tin Mining Company) first found tin on Cape mountain. Since then numerous other strikes have been made, 10 to 30 miles away. The "granite" of the neighborhood seems to carry tin, sometimes in large crystals, and often intimately mixed with the rock.

[1]U. S. Geological Survey, Bulletin 260, 1905.

THE PRINCIPAL TIN SUPPLIES OF THE WORLD.
(In tons of 2240 lb.)

	1898.	1899.	1900.	1901.	1902.	1903.	1904.	1905.
English production....................(a)	4,684	4,013	4,268	4,566	4,392	4,282	4,132	3,857
Straits to Europe and America........(b)	43,350	44,460	46,058	50,339	51,831	52,212	57,566	56,847
Straits to India and China............(c)	2,551	1,484	1,785	2,655	1,882	3,123	3,261	1,700
Australia to Europe and America..... (c)	2,420	3,337	3,235	3,345	3,199	4,934	4,846	5,028
Banka sales in Holland...............(c)	9,038	9,066	12,631	14,978	14,978	15,070	11,363	9,960
Billiton sales in Java and Holland.....(c)	5,342	5,057	5,882	4,387	3,897	3,650	3,215	2,715
Bolivian arrivals on Continent........(b)	1,000	813	1,900	9,670	10,150	9,790	11,867	12,500
Bolivian arrivals in England..........(b)	3,464	3,940	5,065					
Totals in long tons..................	71,763	72,557	80,824	89,940	90,329	93,061	96,250	92,607
Totals in metric tons................	72,911	73,718	82,117	91,379	91,774	94,550	97,790	94,089

(a) From the British Blue Book. (b) From the circular of W. T. Sargant & Sons. (c) From the circular of Ricard & Freiwald.

TIN MINING IN FOREIGN COUNTRIES.

Australia. (By F. S. Mance.)—The improvement in the tin mining industry in 1905 was very noticeable. In Queensland, the developments on the Walsh and Tinaroo field were of a satisfactory character. The principal mine is the Vulcan at Irvinebank. At the 600-ft. level, the ore in one stope assayed 13 per cent. tin, and yielded over $30,000 a fathom. The lode has been proved to a depth of 900 ft. At Stannary Hills, the mines yielded well, and recent developments point to a substantial increase in the output during 1906. At Stanthorpe, on the border of New South Wales and Queensland, tin ore in large quantity has been recovered by the dredges. The plants operating at Tingha (N. S. W.) also secured large yields. The amount of tin ore recovered by the dredgers at this place in 1905 was 468 tons, valued at $214,747. The alluvial deposits in New South Wales continued to afford profitable employment for a large number of men, and good results were recorded. In the Mount Bischoff, Tasmania possesses the principal tin-mine in Australia; up to the end of 1905 this mine had distributed dividends to the amount of over $10,040,000. The company has taken advantage of the high price of tin to operate on the large bodies of low-grade material, which has an average content of about 1.25 per cent. tin. The cost of mining and treatment is covered by $2.88 per ton, and, as shown by the results, this has left an excellent margin of profit. The Briseis and New Brothers Home mines reaped the benefit of the extensive development work carried out in previous years. A well augmented output was consequently shown and the Briseis company was enabled to liquidate its overdraft, which at the commencement of 1905 stood at $200,000. As in the other States, the dredging for tin in Tasmania was attended with encouraging results.

The tin and tin ore exported from New South Wales in 1905 was valued at £173,806, which was £14,571 less than in 1904. However, the production for 1904 was the largest for the previous 14 years, and, excepting 1904, the yield for 1905 is the best since 1890. As previously mentioned the dredges contributed largely toward the yield, the value of the tin ore won

by these plants during 1905 being £50,904. Both in the Tingha and Emmaville divisions stream-tin in large quantity has been won, and it is surprising how well the alluvial deposits, which have been extensively worked for so many years, still continue to yield. It is noticed that the workings are being extended into deeper ground with satisfactory results. The total output of tin ore by the dredges in 1905 was 532 tons, valued at £50,904, as compared with 319 tons, valued at £26,180, for 1904. In the Tingha division there are 15 dredges in operation. The number of dredging plants in commission in the State at the end of 1905 was 49, valued at £264,934. Very close attention is being devoted to this branch of the industry, and the indications favor a much larger output in 1906. The total value of the output from the tin-fields of this State to end of 1905 is £7,436,461.

New South Wales.—The working for tin at Tingha is reported to have from six to ten years' work in front of it. The development of some of the lodes and deep leads of the district is urged. According to E. F. Pittman, the geological conditions of the district are almost precisely similar to those of Cornwall.

South Australia.—The report of the Stannary Hills Mines and Tramway Company for the six months ended Sept. 30, 1905, showed production of 8445 tons of ore, averaging 3.12 per cent. tin oxide, yielding 232 tons of concentrate, averaging 71.2 per cent. tin, as against, for the six months ended March 31, 1905, 7409 tons, averaging 4.7 per cent. oxide, producing 305 tons concentrate, averaging 71.91 per cent. tin. The average grade of the ore was materially reduced, owing to the old low grade dumps having been treated during the latter half-year. When a full supply of water is available the manager estimates that he will be able to supply and treat at least 2500 tons of ore monthly.

Queensland.—An interesting occurrence of tin and bismuth ore has been described.[1] The Lancelot lode at Silver Valley (Herberton district, Queensland) is a true fissure vein, striking about 55 deg. west by north, and dipping south 73 deg. The outcrop is traceable on the surface for over 2000 ft. as a copper lode, and has been worked at shallow levels for this metal in several places. The copper ore is payable only where secondary concentrations of oxidized ores have accumulated. Small quantities of tin are usually associated with the copper, but except in the vicinity of the present workings the lode does not carry payable tin at the surface. It is thought highly probable that tin will be found below the copper when the lode is opened up in depth. This has been found to be the case with similar lodes in Cornwall and elsewhere, and indications obtained up to date at the Lancelot are in favor of this theory.

[1]Abstract of a description in the Annual Report of the Under Secretary for Mines, Queensland, 1904.

The present tin workings are situate at the northwest end of the lode outcrops. To the northwest of the workings, the old slates and sandstones in which the lode occurs are overlaid by conglomerates of younger age than the lode. It is therefore impossible to say at present how far the lode extends in this direction. The lode consists essentially of quartz, a good deal of iron and copper pyrites, with tin oxide and metallic bismuth as the minerals of commercial value. Both the tin and bismuth are found not only in the quartz vein-stone, but also to a considerable extent as an impregnation of the wall rock (slate). It often happens that, although the vein-stone proper is but 12 or 14 in. wide, it pays to take out and send to the battery "formation" for a width of from 3 to 6 ft. The average width of the payable stone is from 2 to 3 ft. The tin content varies considerably. The richer shoots carry from 15 to 20 per cent. of tin oxide, and there has been as much as 3 or 4 ft. of this class of stone showing in the stopes. Usually there is only a small seam of it, while the rest of the lode-matter may assay 4 to 8 per cent. of tin oxide.

During 1904 there was crushed 3319 tons of stone, yielding 198 tons of tin-bismuth concentrate, assaying on an average 57 per cent. tin and 3½ per cent. bismuth. This gives an average of 6 per cent. marketable ore in the stone, not counting loss in dressing, which is considerable. The poorest stone sent to the battery carries from 3 to 4 per cent. tin-bismuth ore. The tin-bismuth concentrate is shipped to Europe.

A newly developed tin mine in the Tinaroo district was described in the London *Mining Journal* of Oct. 21, 1905. The Vulcan and Stannary Hill mines were described in the same journal, May 13, 1905.

Tasmania.—The Blue Tier is a granite range in the northeastern part of Tasmania, between the Ringarooma river and the east coast line. It has long been celebrated for its extensive low-grade tin deposits. According to W. H. Twelvetrees, the Government geologist, who investigated the district in 1901, the Tier granite, with textural and mineralogical variations, is the basement rock of the whole east coast of Tasmania. At the Tier it is a rather coarse-grained porphyritic granite, with large crystals of feldspar—gray, white or pink in color—scattered through it. Its mineral constituents are feldspar, quartz, biotite and a little muscovite. Sometimes the white mica is quite absent; its abundance seems to depend upon the presence of tin ore.

The quality of the tin ore from the Blue Tier mines is good. The ore is remarkably free from iron; in some of it, however, traces of copper are present. There seems to be no rule for the distribution of tin ore in the granite. The patches are irregular, and will, probably, be found to be more dependent upon the natural fissuring of the rock than upon any law of segregation. Taking the formation as a whole, the probabilities are that the ore will average about 0.25 per cent. cassiterite or say 3.5 lb. metallic tin.

The principal companies operating at present are the Anchor and the Australian. Attention is being given to the field by the decision of the directors of the Mount Lyell company to secure leases on tin properties in the Blue Tier and test them with a view to operation on a large scale. At Haley's lease work done in the past exposed a zone of stanniferous rock from one to three chains wide. According to Mr. Twelvetrees this run of tin-stone is connected with a formation worked to the south in the Australian sections, and he considers that there is evidence of such extension into the Full Moon property, which lies some distance to the northeast. There is another run of tin granite formation, extending from Haley's lease, which has been worked at Beale's claim at Mount Michael. This runs north and south on Beale's lease. The ground has been operated on principally between Mount Michael and Little Mount Michael, and, according to Mr. Twelvetrees, the tin-stone proved is wider than on Haley's lease. The Full Moon properties lie to the northeast of Haley's. The tin-stone on the claim is believed to be the richest on the Blue Tier.

The manager of the Anchor mine, in a recent report, while explaining its low working cost, points out that the tin-stone in the formation is of such uneven distribution that prospecting bores do little more than roughly indicate the value of the ground bored. He mentions, further, that the value of the ore in the deeper ground has been less than anticipated, and less than the bores and other tests of the shallower ground indicated.

A company has been formed to develop the tin deposits near Gladstone, through which runs the Mussel Roe deep level. According to an official report, the results of the bores show that nearly 38,000,000 cu. yd. of wash, of an average value of 1.1 lb. of tin oxide, exists. It is intended to provide three modern dredges with buckets of 7 cu. ft. capacity, and capable of raising a total of 150,000 cu. yd. per month. This, it is estimated, can be done at a cost of 2½d. per cu. yd.

On the northeast coast, large claims are being opened up for tin-sluicing purposes. Much of the ground has been worked in the past, but the lower drifts could not be worked by the old-fashioned methods, and it had to be abandoned to the Chinamen to make a living by scratching. But the Pioneer company showed what could be done by up-to-date machinery, and, as a consequence, claims are now being worked in many directions from near Scottsdale to Gladstone and the Scamander. These tin drifts cover a large area of country, and can be treated very cheaply by sluicing or dredging. The latter method has been adopted by the Gladstone company and is giving satisfactory results. Tin-sluicing on the east coast has attracted many men.

There has also been a decided improvement in tin mining in the northwestern and western divisions, the Mount Bischoff mine turning out tin with a regularity which is wonderful. That company has now paid

£2,025,000 in dividends on a capital of £60,000 in 12,000 shares of £5 each, and the mine is said to be looking as well as ever.

The report of the Mount Bischoff Tin Mining Company, for the half-year ending Dec. 30, 1905, showed the following costs for mining and milling per ton of ore: Mining, including new work, maintenance, and other expenses, 3s. 2.908d.; filling, hauling, and emptying trucks, 0s. 5.071d.; crushing, dressing, and maintenance of plant, 0s. 11.455d.; slime sheds, 0s. 1.237d.; ringtail sheds, 0s. 2.359d.; management and supervision, 0s. 9.364d.; plant, including all machinery, 0s. 2.260d.; development and progressive work, 0s. 2.191d. (of this 0s. 0.043d. has been expended on diamond drilling); waterworks, 0s. 0.185d.; ore bagging, 0s. 0.498d.; stores, 0s. 4.301d.; sundries, 0s. 0.798d.; electric light, 0s. 0.557d.; total, 6s. 7.184d. Mineral obtained during the six months, 636 tons; since the formation of the company, 64,775 tons. The quantity of ore smelted on account of the company amounted to 590.5 tons, yielding 413 tons tin. The quantity smelted on behalf of the public was 1593 tons, making the total quantity smelted for the half-year 2183.5 tons. The refined tin produced during the half-year gave an average assay of 99.986 per cent.

Victoria.—The high price of tin has failed to stimulate prospecting in Victoria, although tin ore has been found in several localities. The production of tin ore was an important industry in Victoria over 40 years ago, but now the output is only about 100 tons per year.[1]

E. J. Dunn, director of geological survey, points out that in the county of Delatite, in the Strathbogie district, there is an outcrop of granite which has been proved to be stanniferous. Further south at Buxton, almost in a direct line, the granite outcrops again, and is tin bearing. Continuing the line south, the granite outcrops with tin at Beenak, and then at the most southerly point of Victoria, Wilson's Promontory, the granite and tin outcrops are again met with. The continuation of this rock across Bass' Straits to Tasmania is shown by the chain of islands of similar formation, which, at one time, connected the mainland with the northeast coast of the island.

Thus there is apparently a continuous belt of granite extending from the northeast of Victoria to the northeast of Tasmania. It has been proved to contain tin in various parts of Victoria, where it outcrops, and in Tasmania rich returns have been obtained from it. The Tasmanian portion of the belt is more frequently met with on the surface, and therefore has been more subject to weathering than the Victorian portion. This has resulted in the rich tin washes which enable the northeastern tin companies to obtain the excellent returns reported by them.

Western Australia.—Tin ore has been mined for several years past on the Greenbushes and Pilbarra mining fields, the output being about 200

[1] *Australian Mining Standard.*

tons of tin per annum. Some recent discoveries of tin in the Wodgina district have attracted attention. A. G. Maitland, Government geologist, describes the district as follows:

The Wodgina tinfield is situated on the headwaters of the western branch of the Turner river and within the limits of the Pilbarra goldfield, about 74 miles from Port Headland. Geologically, the field consists of a series of sedimentary and bedded igneous rocks, skirting an extensive granite mass, which occupies a large area of country. These sedimentary beds are much faulted, and have a prevailing dip to the west; they occupy a rugged range, which rises to considerable altitudes above the level of the surrounding plains. The sedimentary rocks are pierced by granite and pegmatite veins (in reality offshoots from the mass previously described), which invariably form the matrices of the tin ores.

Wherever the pegmatitic veins have been opened up, it is invariably found that the tin occurs on either wall of the vein as a band (of more or less width), together with mica and tourmaline in varying proportions, though in one case the occurrence of tin ore in the vein itself was noticed. The bed of the ravines and the slopes on the hillsides carry detrital and residual tin and tantalite everywhere over the whole area occupied by the granite and pegmatite veins. A careful inspection of the surface shows that the tin lodes are numerous, and occupy a considerable area of country; it remains to be proved, however, whether they can be profitably mined, for operations have yet hardly gone beyond the most rudimentary prospecting stages.

Bolivia.—This country is one of the most promising sources of increased tin supply, although its output in 1905 did not show much advance, chiefly because of drought. Difficult transportation is a retardent to the development of these mines, but this is being rapidly improved so far as possible. Many of the mines are situated high in the Andes. Near the town of Santa Barbara (altitude 15,911 ft., probably the highest inhabited place in the world) the principal tin mine is at an elevation of 17,400 ft.

According to a recent report of the British Foreign Office, since the rise in the price of tin the once famous silver mines of Oruro have depended for their prosperity more on their tin output than on the silver. About 50 per cent. of the tin production of Bolivia comes from the Oruro district. In 1904 the total shipments from the district amounted to the equivalent of 8000 tons of metallic tin. In 1903 the output was about half as large.

Miners are scarce and wages have doubled in the last few years; however, many kinds of labor-saving machinery and appliances have been introduced. A comparatively new tin-mining region in the "Tres Cruces" has been opened by Americans, with gratifying results. The Antofagasta & Bolivia Railway Company is considering the advisability of constructing

branch lines to the various tin regions of the locality. At present the mines are generally remote from the railroad.

(By John D. Minchin.)—The tin mines of Bolivia are scattered through the mountian ranges bordering on the eastern side of the Andean tableland, and extend about 300 miles from north to south. The general geological formation is a metamorphic shale, evidence of igneous action being invariably visible in the immediate neighborhood of the deposits.[1]

Probably 95 per cent. of the tin production of the country is from lode mining, only a small amount of stream tin being collected. The importance of the deposits is variable; everything from thin stringers to lodes of 5 or 6 m. width is met. The cassiterite presents itself as solid masses, nodules, and grains, usually in a ferruginous matrix, not unfrequently with iron pyrites and rarely with wolfram.

Ores running as high as 40 per cent. and 50 per cent. of tin are not uncommon, and under favorable conditions as low as 3 per cent. may be worked at a small profit; but the average tin content of ores worked by the larger mines may be estimated at 8 to 10 per cent. Ores worked for silver also frequently contain from 2 to 5 per cent. of tin oxide, which in such cases is cheaply extracted from the tailings.

The treatment of nearly all the ores in the country is limited to ordinary dressing, the product being exported in the form of *barrilla*, containing on an average 64 per cent. of metallic tin. In the Potosi district, owing to higher freights, a quantity of low grade concentrates are reduced in small water-jacket furnaces with charcoal and run into bars.

For reducing the ores, crushers, stamps, and ball and Huntington mills are employed, and occasionally rolls, while the concentration is effected on Wilfley and other tables, and in Cornish round buddles and revolving tables, the old rectangular hand buddles with a broom being still to some extent also employed.

Unfortunately, there is little or no water power available on the Bolivian tableland; consequently steam power is usually employed, llama dung and other native fuels being used; these, however, are of poor quality, and are becoming scarce in many districts, while imported coal costs $39 per ton.

Of late years several anthracite producer-gas engines from the "Deutz" works in Germany have been introduced with satisfactory results as regards economy, the working cost being about 4c. per horse-power hour.

The total production of Bolivia during 1905, reduced to bar tin, may be estimated at 13,000 tons of 2240 lb. I am able to give the exact production in fine tin of the following enterprises, viz.: Compania Minera Huanuni, 1192 tons; Compania Minera Unica, 485 tons; Abelli y Ca. Avicaya, 722 tons. Among the important enterprises are the following: Chorolque

[1] London *Mining Journal*, March 17, 1906.

(F. A. Aramayo), Llallagua (Pastor Saiur), La Salvadora (S. Y. Patiño), Huanuni, Penny & Duncan.

Burma.—Alluvial deposits occur over an extensive area in the valley of the Pakchan river, which is the boundary between Lower Burma and Siam. These are worked by natives at Maliwun, Karithuri and Bopyin· At Maliwun, lodes also are worked. Alluvial deposits also occur in the valley of the Klong Pa Hom river, which empties into the Pakchan river, north of Maliwun. Tin mining in Lower Burma was described by A. B. Snow in the London *Mining Journal,* Sept. 2, 1905.

China.—Tin has been mined for domestic consumption in Yunnan since time immemorial; perhaps also in other parts of China. Recently, alluvial deposits have been developed in the vincinity of Cao-Bang, in the north of Tonkin. On the Tinh-Tucgrant, the yield is said to be 11 ℔. of black tin (50 per cent. Sn) per cu. yd.

Dutch East Indies.—Tin is produced in the islands of Banca, Billiton, and Singkep. The first two are important sources of supply. Their output appears to be, however, decidedly on the wane, which is due largely to shortage in labor supply.

According to the thirty-fourth annual issue of the "Jaarboek van het Mijnwezen," for the year ending with February, 1905, there was a decline in the tin production of these colonies. The figures for the last three years are as follows: 1902–1903, 171,213 piculs; 1903–1904, 185,691 piculs; 1904–1905, 148,987 piculs. The falling off in 1904–05 is accounted for partly by the working year having been slightly shorter than the preceding one, beginning on Feb. 16, 1904, and ending Feb. 3, 1905, with a total of 354 days, this being 29 days shorter than 1903–04. The climatic conditions appear also to have been at times unfavorable, with a scarcity of water for the months of August to October in Muntok and Toboali and other parts, whilst in other districts during December and January floods and torrential rains hindered work considerably. The health of the mining areas, though better than in the two preceding years, was yet not altogether satisfactory. Beri-beri claimed many victims again, especially in the early part of the year, and at Pangkalpinang, although the number of men attacked by the disease was less, it still remained considerable. In the beginning of 1905–6 the epidemic recommenced seriously, specially in the districts of Blingoe, Soengei-Liat and Pangkalpinang. During 1904–05 the total of deaths due to the epidemic was 184 miners, being 1.57 per cent. on a total number of 11,693 men employed.

In 1904, 25,920 piculs of tin were refined at a cost of fl. 9812.26, the loss of tin being 1768 piculs, or 6.82 per cent.; the cost of refining was about fl. 0.40 per picul, against fl. 0.33 for 1903–04 and fl. 0.60 for 1902–03. By the re-working of waste products, amounting to 4537 piculs, a surplus of 2786 piculs of tin was obtained at a cost of fl. 32.15, making the net loss

of tin only 242 piculs. The total cost of production of tin in the Netherlands East Indies for 1904–05 averaged fl. 28.53 per picul, against fl. 28.18 in 1903–04. A further charge for sale in Holland of fl. 3.94 brought the total cost for 1904–05 to fl. 32.48, against fl. 32.31 in the preceding year. The average sale price in Holland having been fl. 94.68, the profit per picul works out at fl. 62.70 for 1904–05, against fl. 61.25 for 1903–04.

Private tin mining is carried on in Billiton and Singkep by the companies bearing those names. In 1904–05, Billiton produced 71,706 piculs, of which 22,936 piculs were in the form of tin ore, sent to Singapore for smelting. In 1903-04, the totals were 64,266 piculs and 15,968 piculs respectively. The total output was increased to 72,672 piculs, or 4326 tons, by resmelting of refuse, and was nearly 9000 piculs better than in the preceding year. The number of men employed was 9732, against 8702 the year before. The Singkep Tin Mining Company produced from July 13, 1903, to June 30, 1904, 4123 piculs, against 6853 in 1902–3. The number of employees was, on an average, 1707 men, of whom about 1400 were mine workers. The output of the mine was valued at fl. 275,052, against fl. 499,337 the former year.

In regard to the Banka tin shortage, according to a recent report (end of 1905), a proposal has been laid before the Java Government for securing at Singapore a supply of coolies to work in the Banka tin mines. In that island not only are working mine coolies scarce from disease, desertion, and departures, but they are also terrorized over by the so-called "Three Fingers" Secret Society. The latter does not scruple at murder and shows such strength that the authorities are powerless.

France.—M. Guedras reported the discovery of a small vein of tin in the commune of Barjac, in the Department of the Lozère. The vein is 2.3 m. wide, of which 0.4 m. is barium sulphate, that mineral, together with quartz, being the principal gangue. Between the barytes and the quartz there is a veinlet of cassiterite, about 2 cm. wide, mixed with pyrolusite and wolframite. The discovery is probably unimportant.

French Indo-China.—Tin deposits in the province of Laos have been described by L. Gascuel (*Annales des Mines*, 1905, p. 321). They are remote, of low grade, and appear to be of no commercial promise.

Great Britain.—The production of tin in Cornwall in 1905 was 3857 tons.

The Dolcoath Tin Mining Company in 1905 crushed 48,472 long tons of ore, yielding 866 tons of black tin, being at the rate of just over 40 lb. per ton of crude ore. The average price obtained for the black tin was £91 18s. The amount realized during the second half of 1905 was £79,000, and the net profit £22,000. Developments continue to be good and at one part of the mine, at a depth of nearly 3000 ft., the ore is opening up extremely well, so that the general prospects of the company are distinctly hopeful.

Carnbrea and Tincroft Mines, Ltd., works ore of lower grade than the Dolcoath, but it produces also some copper, arsenic, and wolfram. During the last half of 1905 29,600 tons of ore were crushed, yielding 496 tons of black tin, or 30.72 lb. per ton. This yield is slightly higher than in previous years. The average price realized was £84 per ton. The receipts from all sources were £34,000, and the net profits were £6690. At these mines also the developments continue to expose richer ores; so that, as in the case of Dolcoath, the present outlook is favorable.

Indo-China.—Tin has been obtained in the Chinese Province of Yunnan for many years, most of it never reaching the outside world. More recently, alluvial deposits have been opened in the vicinity of Cao-Bang, in the north of Tonkin and just south of the Chinese boundary. They lie in a high valley on Pia-Ouac mountain, where a granulite encloses veins carrying cassiterite and wolframite. The Tinh-Tuc grant is the most actively developed; here the alluvium, upon washing, is said to yield 5 kg. of cassiterite per cubic meter, the concentrate averaging about 50 per cent. tin. Sluice boxes of the Malacca type are used, with which the loss of fine ore is extreme. At a few workings the ore is reduced in crude furnaces using wood as fuel, with primitive blowing devices, in spite of which the yield reaches 80 per cent. of the assay contents of the ore. The product is of good grade, owing to the absence of wolframite and mispickel in the concentrate; it is run into 25-kg. pigs, which are sold to traders at 60 to 70c. per kilogram. About 1 kg. per day is the average of the coolie laborers.

Mexico.—Some development work was done on tin mines about 12 km. from Paso de Sotos, in the state of Jalisco, Aguascalientes, 35 miles distant, being the nearest shipping point. Some explorations were made also in the Jeres district, state of Zacatecas at the San Juan Bautista mine. Judging from the accounts, however, these mines have the same characteristics as other tin mines that have been worked in Mexico, i. e., small stringers and *guijilos* of rich ore, but only in small quantities, and without any evidence of a continuous deposit. The occurrence of tin in rhyolite tuff, and similar volcanic rock, is widespread in Mexico, but so far as present prospects are concerned this country should not be regarded seriously as a probable source of tin in commercial quantity.

Peru.—A small production of tin is made in Peru. The treatment of a tin-bismuth ore at the mines of Huayna Potosi was described in the *Boletin* de la Sociedad de Ingenieros, of Lima, May, 1905.

Straits Settlements.—In the Federated Malay States, the Chinese are the chief producers of tin. In shallow workings, despite their wasteful methods, they can still make profits, but leading Chinese miners say that after tin falls below $70 a picul a large number of mines must close down. According to F. D. Osborne, manager of the New Gopeng company, tin is

daily becoming more difficult to get in the Straits, the richer deposits having been exhausted, forcing attention to the lower grade propositions, and these can only be dealt with by machinery, which will enable large quantities to be turned out at a reduced cost.

In Perak, in 1905, the district of Kampar was the only one which gave good returns. The major part of the output came from the large mines, as Tronoh, Gopeng, and those of the French Mining Company. The output of the small miners was comparatively unimportant. The country is undergoing an economic change in its conditions. The shallow mines are being exhausted; the price of food has risen 20 per cent. during the last two years; and the wages of coolies have risen in the same proportion. Not only have wages risen, but the quality of the food required has risen also. Machinery and labor-saving appliances are becoming more and more essential to success, and the time of coolie labor and surface working is passing away.

PRODUCTION OF TIN IN THE MALAY FEDERATED STATES.
(In piculs of 133⅓ lb.)

	1900.	1901.	1902.	1903.	1904.	1905.
Perak	355,590	385,060	405,870	436,296	443,507	446,781
Selangor	269,490	302,570	278,360	284,592	300,413	289,867
Negri Sembilan	82,320	75,230	73,520	85,461	84,849	85,133
Pahang	15,700	26,310	23,120	25,317	27,469	34,879
Total in piculs	723,100	789,170	780,870	831,666	856,238	856,660
Total in metric tons	43,123	47,713	47,211	50,254	51,790	51,793

Johore is an independent State under the rule of a Malay sultan, in the close neighborhood of Singapore, which has attained some prominence as a tin producer. Its area is about 9000 square miles, with a population of about 250,000, principally Chinese. The population is confined to the districts near the coast and large rivers, the interior being covered by virgin forests, not yet explored.

Alluvial tin mining is carried on by Chinese in many parts of the State, but owing to primitive methods of working, the output is not large. The Royal Johore Tin Mining Company, Ltd., is the only European company in Johore. During the last seven years it has met with fair success, turning out about 2000 tons of tin concentrate, giving on an average 74 per cent. tin. In many places the assay is as high as 76.5 per cent. The richest deposits are found at the foot of mountains, in the old silted-up beds of creeks. Top soil averages 20 ft. and wash dirt 2 ft. The latter frequently gives 4 to 8 lb. oxide to the cubic yard, which pays well, even with the slow and out-of-date methods of mining. The mining on the company's property is all alluvial; up to the present no lodes have been located. The company clears the ground of forests, makes dams and drains and then lets the ground out on tribute to Chinese, who pay from 25 to 30 per cent. of their output; 8 per cent. of this, however, goes to the

Government. A paddock 200x900 ft. will be opened up by about 600 Chinese, in gangs of 50 to 60 men, and if they can get 1 lb. per man per day of black tin they are quite satisfied.

Tringganu is an independent native state on the east coast of Malay Peninsula, lying in a measure between Pahang and Kelantan (a Siamese-Malay state). Tin mining is carried on in Kemaman (the southern half of the State). Running more or less parallel with the coast, and at about 20 miles distant from it, is a range of hills forming the divide between Tringganu and Pahang. The foot-hills and the plains on their eastern flank are the seat of the tin mining industry.

Geologically, these hills are all of one type, their essential features being a core of granite capped by much-altered, tilted, and shattered beds of shales and grits. On the lower slopes, and in the flats beyond, hardly a trace of the sedimentary beds remains *in situ*. The alluvial soils and gravels, nowhere of great depth, rest immediately on a granite bedrock. Everywhere these gravels carry stream tin, and, in many cases, the shallowness of the deposit is more than atoned for by the extreme richness of the ground.[1]

The most striking feature is the extent of the old alluvial workings of the Chinese. That any readily accesible alluvial tin deposits have been left is not to be expected. But where either the lack of water, or the presence of too much water, is encountered, a prosperous future is still in store for enterprises directed by competent engineers.

An interesting and valuable feature of these alluvial fields is the occurrence of the minerals affording rarer metals, especially monazite and xenotime. On one concession being worked for tin by Chinese tributers, fully 50 per cent. of the material collected in the sluices was monazite; and in another instance, xenotime could be abundantly gathered (with zircon, monazite, cassiterite, ilmenite, etc.) from pot-holes in the granite beds of the streams. These properties are owned by wealthy Singapore merchants, and are likely to be worked on commercial lines.

At varying elevations in the foot-hills occur lodes of tin-stone, from the eroded portions of which the alluvials have been formed. So far as they have been anywhere followed, they do not in any instance live as payable orebodies in the granite itself. Nevertheless, that their genesis is due to the granite cannot be doubted. Tiny stanniferous leaders are found in the granite, though not in association with, nor even in close proximity to, any workable tin lode. On the other hand, extremely fine grained cassiterite, widely disseminated through the granite, is occasionally found. The only profitable orebodies hitherto found are in the shales overlying the granite. But though these are confined to the shales, they are never at any great distance from the main granite

[1]. C. G. Warnford Lock, *Proceedings*, Institution of Mining and Metallurgy, 1905.

core, and are often closely associated with bunches and bands of granite, which, however, would seem to be always of a more porphyritic character than the main mass, and much more prone to decomposition.

Swaziland.—Parcels of tin ore from this source continue to arrive at Swansea, but the quantity is small.

Transvaal.—Great interest was excited in South Africa by the discovery of tin in the Bushveld area, northeast of Pretoria, which led to rampant speculation and a sad collapse. A lode in granite, carrying a fair percentage of tin, was located early in 1904, and the Bushveld Tin Mines, Ltd., was formed to operate it. The mine has been opened to a depth of 200 ft. and is reported to be showing well in the bottom. A mill has been erected, which will begin crushing early in 1906.

Rich ore discovered in May, 1905, to the northeast of Bushveld caused great excitement. The property was examined by W. Frecheville, who reported that it had fair prospects. Exploratory work, however, appears to have been poorly directed.

The occurrence of tin ore in South Africa was described in *South African Mines* of Apr. 22, May 6, and Aug. 19, 1905.

The Tin Markets in 1905.

New York.—The statistical position of tin, which during 1904 was supposed to be strong, really proved to be so to a very marked extent. How far this is true of the United States is best illustrated by the fact that although the imports during 1905 were more than 4000 tons larger than during 1904, the quantities of tin held in stock were smaller at the end of the year than they were at the beginning. In consequence of this fact it is not to be wondered at that prices reached the highest level for a generation.

At the beginning of 1905, Straits tin was selling at about 29½c. per lb., which price remained stationary for futures until the middle of March, while spot tin fluctuated in sympathy with the stocks that were available, commanding as high as 30c. Toward the end of March, London quotations advanced considerably, and buyers here were compelled to follow, with the result that about the middle of April quotations were 31c. for spot, and 30c. for futures. During the latter part of April prices declined to 29¾@30c. per lb. for spot, and remained at about this level until the middle of June.

From this period onward the market advanced gradually in sympathy with London, where speculation reigned supreme. Bull operators were assisted in July by the announcement that Banka sales would be further reduced, and a price of 33c. per lb. established. Profit taking, which continued during August and September, weakened the market temporarily and by the middle of October prices here had receded to 32c. per lb

Fresh inquiry developed at these lower prices, and the demand became extraordinary. Quotations jumped by leaps and bounds, and reached 35c. by the beginning of December.

The consumption continued at an extraordinary rate during the month of December, and as shipments from abroad had decreased considerably, owing to the fact that there was no spot tin available in London from the Straits, and stocks could only be replenished by shipments via London, the spot market in particular became extremely firm, and a premium was readily paid for metal on the spot. The market at the end of this year was quoted at 36@36¼c., depending upon deliveries.

AVERAGE MONTHLY PRICES OF TIN PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1896	13.02	13.44	13.30	13.34	13.54	13.59	13.63	13.49	13.15	12.94	13.09	12.96	13.29
1897	13.44	13.59	13.43	13.34	13.44	13.77	13.89	13.80	13.98	13.88	13.79	13.71	13.67
1898	13.87	14.08	14.38	14.60	14.52	15.22	15.60	16.23	16.03	17.42	18.20	18.30	15.70
1899	22.48	24.20	23.82	24.98	25.76	25.85	29.63	31.53	32.74	31.99	28.51	25.88	25.12
1900	27.07	30.58	32.90	30.90	29.37	30.50	33.10	31.28	29.42	28.54	28.25	26.94	29.90
1901	26.51	26.68	26.03	25.93	27.12	28.60	27.85	26.78	25.31	26.62	26.67	24.36	26.54
1902	23.54	24.07	26.32	27.77	29.85	29.36	28.38	28.23	26.60	26.07	25.68	25.68	26.79
1903	28.23	29.43	30.15	29.81	29.51	28.34	27.68	28.29	26.77	25.92	25.42	27.41	28.09
1904	28.85	28.09	28.32	28.13	27.72	26.32	26.57	27.01	27.78	28.60	29.18	29.29	27.99
1905	29.325	29.262	29.523	30.525	30.049	30.329	31.760	32.866	32.095	32.481	33.443	35.835	31.358

London.—The beginning of 1905 found this market well maintained and well able to resist the attacks of "bear" sellers, whose operations for a fall had been latterly rather aggressive. Stocks on the European continent were very light. Eastern sellers, however, who had hitherto offered with reluctance, opened with rather free selling at receding prices. Spot warrants, which had opened at about £134, fell toward the middle of the month to £131 under pressure of heavy realizations and bear sales. The month closed with cash warrants at £132 7s. 6d. and three months' at £131 2s. 6d.

February opened with a fairly brisk market, but before the middle of the month activity subsided and prices drifted downward to £130 7s. 6d. for cash, and £129 10s. for three months. The month closed with a steady trade; cash warrants at £129 17s. 6d., and three months' at £129 12s. 6d.

March was comparatively uneventful until the second week, when the Dutch Government unexpectedly announced that the quantities of Banka tin to be disposed of by auction during the remainder of the year would be reduced by 1700 tons. This was due to a diminution of production owing to scarcity of labor. The result was a sharp upward movement, in which consumers and speculators alike participated, until £136 10s. was paid for spot warrants. Then followed a temporary relapse to £135 10s.; but renewed buying carried cash tin to £138 10s., and three months' to £134 10s. at the end of the month.

April continued the upward movement, which culminated in £145

paid for early delivery. Forward metal, being less firmly controlled, did not rise above £136 10s. With the liquidation of bear sales, the heavy premium on cash warrants abated, and the end of the month the cash price receded to £138 5s., the three months' price being £134 5s.

May opened with favorable trade reports, but prices fluctuated daily, mostly in decline, until about the middle of the month, when cash warrants commanded only £135 5s. Then an improvement set in, prompted by an official announcement that the Banka production for 1904-1905 amounted to only 7700 tons as compared with 12,438 tons during the previous 12 months. This caused cash tin to rise to £137 10s. and three months' to £136 2s. 6d. Late in the month the cash price fell to £135 15s., and the three months' to £134 12s. 6d. However, the Banka auction price was equivalent to £139; and the result was a quick recovery in the London prices, which closed at £136 5s. for cash, and £135 5s. for three months'.

June, at the opening, was comparatively uneventful, with restricted operations and with unimportant fluctuations. Statistics continued favorable for a prospective rise, one noteworthy feature being a falling off of 1500 tons in the supply from Bolivia during the first five months of the year. The middle of the month brought about an improved demand and the cash price which, at the beginning of the month, had stood at £136 5s., advanced to £140 5s. The month closed with cash warrants at £139 10s., and three months' £138 5s.

July opened with an initial decline of 15s., but this was quickly recovered and the market rose to £144 5s. Thereafter speculative buying subsided in view of the Banka auction due toward the end of the month; and the cash price was allowed to fall back to £142 7s. 6d. But a surprise was in store. It was announced that sales of Billiton tin during next year would be reduced by about 1400 tons, or practically 50 per cent. This caused the cash price to touch £145 10s. at the bound. Then came another startling announcement from the Dutch Colonial Minister, that sales of Banka tin during 1906 would have to undergo a further reduction of about 1500 tons. This caused a rush of buyers, who quickly carried the price to £150 10s. for cash, and £150 for three months. The usual reaction followed, and the month closed with a very sensitive market at £149 5s. for cash. and £148 2s. 6d. for three months.

August saw the market relapse into dulness. Some small bear sales served to depress prices to £149 for cash, and £147 5s. for three months'; but consumers were alert to take advantage of this move, and prices quickly recovered. Later in the month cash warrants touched £153. A reaction followed, and the closing prices were £149 10s. for cash, and £148 15s. for three months'.

September opened with large sales for forward delivery at declining prices, £147 5s. being accepted for cash warrants, and £146 10s. for three

months'. Thereafter fluctuations were wide and numerous, and mostly downward, until cash touched £145 10s., and three months' £144 15s. The market was steady at the end of the month, with cash warrants at £146 15s. and three months' at £145 15s.

October found a hopeful sentiment prevailing, and prices rose early in the month to £148 15s. for cash, and £147 15s. for three months'. This improvement was early dissipated by a message from Batavia to the effect that the output from the Banka mines had lately increased and would enable the Dutch Government to revert to its original intention of selling 9500 tons during the ensuing year. This caused a relapse to £145 10s. for early dates, and £144 10s. for three months'. Further decline was arrested by large buying orders, and the second half of the month saw a steady advance. Prices at one time touched £149 5s. for near delivery. There was then a slight relapse; but the month closed with cash warrants at £150 and three months' at £149.

November opened with a strong market, spot warrants fetching up to £150. The high prices were justified by the statistics, which revealed almost the smallest visible supply on record. By the middle of the month the cash price rose to £152 17s. 6d., and three months' to £151 15s. Competition at the Banka sale at the end of the month was exceedingly keen and buyers had to pay over £160 per ton. The London prices at the close were £156 15s. for early dates, and £156 2s. 6d. for three months'.

December found the demand for cash warrants was enormous, so that within a few days the premium on these was over £3 per ton. Prices advanced daily until, on Dec. 15, £166 5s. was paid for spot warrants. Thenceforward sellers showed themselves occasionally aggressive, but their offerings were readily absorbed, and fluctuations continued within narrow limits until just at the close, when sales were pressed in a quiet market, resulting in an unexpected fall from £164 15s. to £162 10s. for cash, with three months' depressed to £161 10d. This decline was explained by pressing offers from Eastern merchants who, anxious to profit by the rising tendency of the Singapore exchange, were inclined to sell their production ahead.

Progress in the Metallurgy of Tin.

Mechanical Concentration.

Tin-Bearing Pyrites.—The stanniferous pyritic ore of the North Dundas district, Tasmania, has been investigated by J. D. Millen, chief metallurgist to the Mount Bischoff Tin Mining Company, whose experiments promise commercial success. Some of the pyrite ore is rich in tin. The process decided upon consists first of calcination, followed by concentration, when, if it is found that the whole of the sulphur has not been removed,

a second roasting will be resorted to. The product will then be smelted in the usual way.

Tin-Wolfram-Copper Ore.—Ore of this character has been successfully treated by washing and magnetic separation at the Clitters United mine, Cornwall. The process has been described in a long paper by F. Dietzsch (designer of the works) in the *Proceedings* of the Institution of Mining and Metallurgy, 1905.

Jigging.—Jigs are used for tin dressing only in Australia; in Cornwall, and elsewhere, the Frue vanner and other forms of the shaking-table have replaced the jig.

Smelting, Etc.

Tin-lead Smelting.—An improved method, due to Carl A. L. W. Witter, of Hamburg, Germany, is now being introduced in the Straits Settlements which renders it possible to keep the tin content of the slag as low as 0.5 per cent. The tin slags are smelted with galena in a reverberatory furnace. An alloy of tin and lead is obtained. As the tin slags contain considerable iron, the sulphur in the lead ore serves to make a matte of it; the tin-and-lead alloy is drawn off from underneath.

A similar method has been described by L. Parry, of Huddersfield, England, who states, in British patent 7154, Apr. 5, 1905, that "foul" tin slag, i. e., slag containing 3 per cent. or more of tin, can be "cleaned" by smelting it in a water-jacketed blast furnace such as is used for lead smelting, the charge being composed as follows: (1) Foul tin slag; (2) iron oxide in such quantity as will give a slag containing more than 40 per cent. of ferrous oxide; (3) lead ashes, slags, drosses or oxidized lead ores in such proportion as to give an alloy containing about 20 per cent. of tin; (4) a source of arsenic or sulphur, such as mispickel, arsenical drosses, spent oxide, or lead sulphate, enough being used to form an easily fusible compound with any metallic iron reduced; (5) a mixture of gas coke and furnace coke in such proportion as to obtain the maximum output, gas coke increasing the reducing action, while furnace coke increases the speed of smelting. The slag produced in this way contains only $\frac{3}{4}$ to $1\frac{1}{2}$ per cent. of tin.

In a subsequent patent (U. S., No. 801,820, Oct. 10, 1905) Witter describes the process of extracting the tin from the tin-lead alloys. The alloy is smelted in a reverberatory furnance, and when the molten metal shows a red glow it is subjected to a blast of air. This oxidizes the tin, and a small portion of the lead. An oxide mixture containing a high percentage of tin can be drawn off the surface of the metal bath, and by repeating the process an alloy very rich in tin is secured.

Hydrometallurgy of Tin.—According to G. Kroupa, in the *Oest. Zeit.*, 1905, Peruvian tin comes into the market in an impure form, and for many

purposes must be refined. This is done by remelting the metal and granulating it. The granules are treated with hydrochloric acid, in which treatment an excess of tin must be present. The tin is dissolved as chloride, while the wolfram, which forms the chief impurity, remains behind in the black residue. The clear solution is further treated by tin, whereby arsenic and antimony are precipitated. If there be any lead present in the solution, this is precipitated as sulphate by the addition of sulphuric acid. From the purified solution, the tin is precipitated by means of zinc, after which the precipitate is washed with acid and finally with water, dried, melted in an iron pot, and cast into bars. The solution from which the tin has been precipitated contains zinc chloride, which is thrown down as oxide by the addition of milk of lime, free from iron. The zinc oxide is filtered off, washed, dried, and glowed, producing a tolerably pure zinc white, which can be used as a pigment.

Electrolytic Tin Refining.—Dr. H. Mennicke, in *Elektrochemische Zeitschrift,* December, 1905, shows that the electrolytic refining of "work-tin" with a tin fluosilicate solution would be practicable and profitable. Among the advantages are the preparation of chemically pure tin, which commands a high price and is especially suited for making tin salts, tin oxide, etc. His conclusions are as follows: (1) The preparation of the tin electrolyte is much more difficult than that of the corresponding lead solution. (2) The removal of hydrofluoric acid from the solution is not necessary. (3) The regeneration and purification of electrolyte is more difficult than in the Betts lead process. (4) The current yield is lower than in the Betts process. (5) Solid refined tin can be obtained under certain conditions. (6) The electrolytic refining of tin is economical in the absence of much lead. The removal of copper, bismuth and antimony is easy. (7) The extraction of tin from lead-tin alloys is possible, but not economical. (8) A combination of electrochemical and chemical processes for tin-lead alloys is not economical.

Electrometallurgy of Tin.—This subject, with especial reference to the detinning of iron scrap, was discussed in a valuable paper by H. Mennicke, in *Zeit f. Elektrochemie,* March 30, 1906, which can not be abstracted satisfactorily.

Miscellaneous.

Tensile Strength of Copper-Tin Alloys.—A. S. Shepherd and G. B. Upton (*Jour. Phys. Chem.*, 1905, IX, 441-476, and *Chem. Centr.*, 1905, II, 538; see also *Jour. Soc. Chem. Ind.*, 1896, 810; 1901, 814; 1905, 241) have tested the tensile strength of copper-tin alloys. They find that the tensile strength of alloys containing the pure alpha-crystal rich in tin is not much affected by heat. Bronzes containing 74.87 per cent. of copper become much stronger if they are heated above 510 deg. C.; by long-continued

heating the tensile strength is diminished, but the ductility is increased. The alloy containing 78.81 per cent. of copper has a maximum tensile strength; it consists of a mixture of alpha- and beta-crystals. It is possible to prepare, on the one hand, a cast bronze with a maximum tensile strength of 60,000 lb. per sq. in. and an extension of 1.5 per cent.; and, on the other hand, a bronze with a tensile strength of 45,000 lb. per sq. in., and an extension of 39 per cent.

Analysis of Tin-Tungsten Ores.—According to *Zeit. f. angew. Chem.*, XIX (1906), p. 140, H. Angenot has shortened somewhat the tedious analytical method for separating tin and tungsten. Experimenting upon known mixtures of tungstic acid and tin oxide, he followed Bornträger's method except that, instead of fusing the substance with sodium carbonate for one hour, he fused it in an iron crucible with sodium peroxide for 15 minutes, and obtained accurate results. The determination of tin was equally satisfactory, and the abbreviated method is now applied directly to the analysis of tin-tungsten minerals.

TUNGSTEN.

By Reginald Meeks.

The remarkable growth in the use of so-called "high-speed" or "self-hardening" steels has caused great activity among producers of tungsten, and prospecting for ores of this metal has proved successful both in this country and abroad. The production of tungsten concentrate in the United States during 1905 amounted to 834 short tons, valued at $257,463 as compared with 740 tons ($184,000) in 1904, 292 tons ($43,639) in 1903 and 184 tons ($33,112) in 1902. No record of imports and exports is available.

The market price is variable, and quotations are not always reliable. First-class concentrate must not contain more than 0.25 per cent. phosphorus, nor more than 0.01 per cent. sulphur and should run 60 per cent. WO_3 or better. In the Boulder, Colo., district, $5 per unit has been freely offered; on this basis it would not be unreasonable to assume the value of the ore in the East to be $5.25 to $5.50 per unit. Early in 1905 it was reported that several large lots of concentrate were disposed of in Germany at prices ranging from $5.36 to $5.83 per unit, while the metal, 96 to 98 per cent. pure, brought 56c. per lb. The demand for tungsten seems to be well in advance of the supply and prices are readily maintained. There is considerable demoralization in the market due to the customs decision allowing the free importation of crude tungsten ore, and certain producers in Boulder county, Colorado, have preferred to shut down their mills rather than take chances with their product.

Tungsten Mining in the United States.

Arizona.—Two miles west of Johnson, Cochise county, are placer deposits of wolframite. The mineral is won by means of rockers and dry washers. A majority of the claims, mostly patented, are operated in a small way by two Eastern firms affiliated with the Primos Chemical Company and the Chrome Steel Works. The tungsten mineral is distributed over several miles of surface, and Mexican laborers working the rockers are paid 10c. per pound for it on the ground. An average workman will wash out 20 lb. or over per day. This concentrate varies in size from a grain of corn to pieces several ounces in weight and from 65 to 75 per cent. wolframite, beside carrying scheelite in considerable quantities. The value of this product f.o.b. Cochise is from 17 to 20c. per pound; shipments of several tons per month are made. The ore also occurs in pure white quartz veins

in the granite, which are easily traced along the surface for several miles. The vein matter will go as high as 30 per cent. wolframite. No active work has yet been done on these ledges, as it has been easier to get returns from the placers and from rich float, which is crushed by hand breakers.

California.—Discoveries of tungsten have brought this State into the list of producers, and deposits of value have caused excitement in the region about Randsburg, Kern county. On the Papoose claim, the ledge is from 18 to 20 in. wide, and the ore is said to assay from 70 to 78 per cent. tungstic acid. It is the intention of the principal tungsten claim-owners to put up a plant on the railroad near St. Elmo to concentrate the ore. The tungsten field, as it is now known, is two by three miles in extent. Ore from this district has been shipped to Germany. Promising deposits have also been located near Johannesberg in the same county.

Colorado.—Boulder county continues to hold the leading position in tungsten mining, and has attained considerable importance. The ore is almost exclusively wolframite, the occurrence of hübnerite and scheelite being rare.

In 1905 the output was between 600 and 700 tons of concentrate, worth locally about $220,000. The ore occurs over a large area centering around Nederland; but most of the deposits are small and bunchy. Toward the end of the year the value of tungsten concentrate fell off somewhat, but even at the lower prices the production was profitable. The tungsten belt seems to be gradually extending and a number of locations of well-defined veins were made, while several properties changed hands.

Tungsten ore has been found at Salina, Wallstreet, Eldora, Gorden Gulch, Caribou and Nederland, but the major part of the work has been carried on at the last three camps.

In San Juan county, hübnerite occurs in solid streaks distinctly separated from the quartz gangue. This permits of a clearer and easier separation than with the wolframite of Boulder county. There are claims on Bonita peak, in Dry gulch, on Sultan mountain, in Burns gulch and near Gladstone. A claim on Bonita peak is at an elevation of over 11,000 ft.

Idaho.—A promising deposit of tungsten is reported in Shoshone county. It has been opened by a tunnel 200 ft. long in the main ledge, and a second one 500 ft. long above the mouth of the first. While no assays have been supplied, it is stated that the ore carries a profitable quantity of tungstic acid and a little gold and silver.

South Dakota.—Near Hill City, Pennington county, a promising deposit of wolframite has been discovered and development has started; a two-compartment shaft has been sunk 50 ft.

Utah.—It is stated that the first tungsten concentrate ever produced in this State in commercial quantity is ready for shipment, having been recovered as a by-product at the Continental Alta mine, Little Cottonwood.

Washington.—In Okanogan county a discovery of what was supposed to be high-grade silver ore was made in 1904, and claims were staked. It was learned that the ore was a good grade of wolframite, the concentrated product yielding about 72.8 per cent. tungstic acid under laboratory tests. Twelve claims have been located by the Tungsten Consolidated Mining and Milling Company, of Loomis, Wash., and are located in three tiers, which occupy almost the entire face of a southerly slope and cover an area of 6000x1800 ft. The ledge in the longer tunnel is 4 ft. wide most of the way, but at 40 ft. in from the portal it is 6 ft. 9 in. The company is driving four tunnels and the veins are showing well.

Tungsten Mining in Foreign Countries.

Australia.—The New England district of New South Wales produces wolframite in economic quantities, although the ore is low-grade and the problem of dressing is a difficult one. The production of scheelite in New South Wales in 1905 was 138.3 long tons (£10,122), as against 15.5 tons (£1,406) in 1904. The production of wolframite during the same years was 86.5 tons (£7,361) and 89 tons (£8,432) respectively. The producing districts of New South Wales are Hillgrove, Emmaville, Uralla Tuena, Barraba, and Tarrington.

New Zealand produces scheelite associated with quartz in a schist country rock. It carries from a few to 18 dwt. of gold per ton. The ore is treated by Huntington mills and Woodbury and Frue vanners, and concentration as high as 72 per cent. WO_3 is accomplished.

Queensland continues to furnish a large proportion of the world's supply of tungsten. In 1901 the production amounted to only 72 long tons, the next year to only 55 tons; in 1903 the output increased to 197 tons and the following year to 1540 tons. In 1905 production amounted to 1413 tons, valued at £100,203.

Brazil.—A group of Hanoverian capitalists interested in the manufacture of tungsten salts has acquired the recently discovered tungsten ore deposits near Porte Alegre, South Brazil. These deposits are reported as of unusual extent and richness, one vein of clean, massive tungsten ore, assaying over 70 per cent. WO_3, having been traced on the surface for over a mile, and averaging fully 1 ft. in width. There are a number of smaller parallel lodes also visible on the surface.

Mill Practice.

In Colorado, ore dressing seems to be carried on along one general line; grizzlies, Blake crushers, stamp batteries, 20-mesh screens, Wilfley tables, hydraulic classifiers and setting tanks being employed. In Australia, Huntington mills replace the stamp battery and vanners are substituted for Wilfley tables.[1]

[1] H. R. Van Wagenen, Bulletin of the Colorado School of Mines.

At the abandoned Clitters United Mines in Cornwall, large quantities of tin-tungsten ore have been picked from the waste heaps and operations have been resumed. The ore is divided, part passing through grizzleys, Blake crushers, stamp batteries and a 25-mesh screen; the smaller part is passed over a shaking screen, through rolls, then vibro screens, ball-mill and finally through a 30-mesh screen.

The resulting pulp is then treated in spitzlutten, spitzkasten, Buss swinging tables, Lührig vanners, distributing boxes, etc. It is claimed that the ball-mill slimes less than stamp mills, so the finer portion of the ore is treated without stamping. Magnetic separators have also given good satisfaction in the separation of wolfram from tin; this method was probably first used at the San Finx mines in Spain.

Metallurgy of Tungsten.

Metallic tungsten is silvery gray in color; difficultly fused; sp. gr. about 19; atomic weight 184. The metal in almost a pure state is prepared by fusing carbonate of soda and tungsten ore, which results in the formation of tungstate of sodium. This is leached with boiling water and the yellow WO_3 is precipitated by hydrochloric acid, dried and reduced to the metallic state in crucibles at high temperature.

M. Defacqz prepared[1] a mixture of 100 parts of tungsten ore from Zinnwald (Bohemia) with 14 parts of coke. It was then treated in a Moissan furnace carrying 950 to 1000 amperes at 50 to 60 volts. The ore analyzed as follows: WO_3, 71.76; SiO_2, 1.69; FeO, 7.60; MnO, 16.30; CaO, 2.28 per cent. On heating 10 to 12 minutes a metallic mass was obtained containing percentages as follows: W, 92.53; Si, 0.49; Fe, 2.37; C, 5.21 per cent. The metal was surrounded by slag, which was easily removed, owing to a slight layer of carbide which deliquesced in air.

In the industrial manufacture of ferro-tungsten a mixture was employed having the ratio of 100 kg. of ore to 250 kg. of English anthracite coal. The treatment took place in an arc furnace carrying 5000 to 6000 amperes at 50 volts. The composition of the ore was: WO_3, 6.92; SiO_2, 2.2; Fe_2O_3, 18.7; Mn, 4.0 per cent. The resultant product was a ferro-tungsten having the following composition: W, 50.99; C, 0.43; Fe, 11.44; P, 0.014; Si, 0.88; Mn, 6.84 per cent. There are three types of ferro-tungsten manufactured: first, high carbon and medium tungsten; second, high carbon and high tungsten; third, low carbon and high tungsten.

Uses.—Tungsten finds its greatest field in the manufacture of steel for special purposes. Alloyed either by itself or in connection with molybdenum and chromium, it imparts to steel great toughness and the ability to resist shock. Hence for high-speed tool steel, armor plate projectiles, car springs,

[1] M. Leon Guillet, in *Le Genie Civil.*

etc., it occupies a prominent position. It is also alloyed with aluminum, copper and many other metals.

Sodium tungstate is used as a mordant in dyeing and for rendering materials, such as cloth, etc., non-inflammable. It is also proposed to utilize tungsten for manufacturing filaments for electric lamps.

Analytical Methods.

O. P. Fritcherle's method for the determination of tungsten in ores is as follows: Weigh into a platinum dish, or crucible, of about 25 c.c. capacity, 0.5 gm. of the finely powdered ore, add equal quantities of hydrochloric acid and hydrofluoric acid, and digest at slow boiling temperature for about one hour, or until the ore is all in solution, adding from time to time more of each acid. Evaporate down to about half the original volume, to displace the silicon fluoride and excess of hydrofluoric acid. The dish should be covered to prevent loss by splattering. The two acids will entirely dissolve most ores, excepting those containing tin oxides, in which case it will be necessary to filter off the insoluble residue.

Transfer the solution to a number three beaker, add 20 c.c. hydrochloric acid and 8 c.c. nitric acid, cover and boil down to about 10 c.c. to expel all fluorine; the tungstic fluoride will be converted into chloride and in the presence of nitric acid will be precipitated as tungstic acid, H_2WO_4. Dilute with 50 c.c. of hot distilled water, and boil slowly for about half an hour or until the tungsten is all precipitated, when the beaker should be set back, and kept just below the boiling point until the solution has cleared. Filter through a Gooch crucible using an asbestos felt, wash well with hot water, dry, ignite at white heat for five minutes, cool and weigh as WO_3.

Zellner describes a volumetric method for use where great accuracy is not required. Fuse the weighed and powdered ore in a platinum crucible with three parts sodium carbonate and one part potassium nitrate; cool, dissolve the melt in hot water and filter. Acidify slightly with acetic acid, dilute to 200 c.c., boil, and while boiling titrate with a standard solution of pure crystallized lead acetate. This consists of 16.3 gm. of the salt, in one liter of water slightly acid with acetic acid. When titrating, a white precipitate forms and settles rapidly on stirring. The end point is the absence of the formation of precipitate and cloudiness.

ZINC.

By W. R. Ingalls.

In making the statistical investigation of the zinc industry for 1905, I have attempted to show the production of smeltable ore, by States, and the distribution of the spelter consumed, according to use. Through the general co-operation of the smelters and most of the consumers it has been possible to present fairly complete statistics on these subjects.

Production of Ore.

Statistics of the production of zinc ore in Missouri and Kansas (Joplin district) and New Jersey are available for a long series of years. Up to a few years ago nearly the whole spelter output of the United States was derived from those sources. In 1899 zinc ore from Colorado began to appear in the market, and during the last two or three years that ore, together with ore from other States and Territories west of the Rocky Mountains, and from British Columbia and Mexico, has been figuring largely in the market.

The ore production of the Joplin district is of two classes, viz., blende and calamine. The former averages about 58 per cent. zinc; in round numbers, two tons of this ore make one ton of spelter. The calamine of the district is entirely zinc silicate. It may be assumed as averaging a little better than 40 per cent. zinc, three tons of ore being required, roughly, to make one ton of spelter. The total production of zinc ore in the Joplin district in 1905 was 252,435 tons. No attempt was made to classify this as blende and calamine, but in recent years the output of the latter class of ore has amounted to 10,000–16,000 tons per annum, and it may be reasonably assumed that the production in 1905 was something between those figures.

A small amount of calamine, both carbonate and silicate, is produced in southeastern Missouri, especially by the Valle mines. This ore goes chiefly to St. Louis, and amounts to 3000–6000 tons per annum.

The zinc ore produced in the States and Territories west of the Rocky Mountains is both blende and calamine, the latter being chiefly zinc carbonate, produced in Mexico and New Mexico. The production of Colorado, Utah, Idaho, Montana, and British Columbia is chiefly, if not entirely, blende. This ore varies generally in grade from 30 per cent. to 50 per cent. zinc. In a few cases, as at Creede, Colo., and the output of hand-sorted, lump ore of one mine in British Columbia, it exceeds 50 per cent., the Creede

ore (mill-concentrate) in fact being almost as high in zinc as the average Joplin product, but although low in iron it is higher in lead than the Joplin ore. The average zinc content of the Western ore, both blende and calamine, may be assumed at 38 per cent. From 3½ to 3 tons of this ore are required to produce one ton of spelter. This sulphide ore is comparatively high in iron and lead; some of it is very high in those elements. It is mostly produced as a concentrate from mixed sulphides, the lead product being shipped to the silver-lead smelters. The Iron Silver Mining Company, however, ships a good deal of hand-sorted lump ore from its Moyer mine, at Leadville.

Wisconsin produces a blende concentrate which, after magnetic separation, is practically as high in zinc as the average Joplin ore, and when well prepared is comparatively low in iron and lead, the blende itself being only slightly ferruginous and the iron content of the marketed ore being chiefly intermixed marcasite. Wisconsin also produces carbonate ore, which is used at Mineral Point for the manufacture of zinc oxide.

The large output of zinc ore in New Jersey is entirely from the Franklin mine of the New Jersey Zinc Company. It is the mixed franklinite-willemite, averaging about 20 per cent. in zinc, which is separated into one product (willemite) for spelter manufacture and another product (franklinite) for manufacture of zinc oxide and spiegeleisen.

Of the Western zinc-mining districts, the most important single district is Leadville, Colo. Other important districts are Creede, Colo., Magdalena, N. M., Park City and Frisco, Utah, Monterey, and Las Plomosas (near San Sostenes, on the K. C. M. & O. R. R., Chihuahua), Mexico, and the Slocan, British Columbia. Outside of these districts, the zinc ore production west of the Rocky Mountains comes from many scattered localities. In New Mexico, besides Magdalena, Hanover is a small producer, and there are several other promising districts. In Montana, Butte is the principal source. In Idaho, the Wood River district is the most important, although some ore was obtained in 1905 from the Cœur d'Alene. In Utah, the Daly-West Mining Company, of Park City, and the Horn Silver Mining Company of Frisco have large zinc resources; the former did not produce in 1905, but the latter shipped 8445 tons. In Colorado, besides Leadville and Creede, zinc ore is produced at Rico, and by many small mines in Clear Creek and Summit counties. In Mexico the Calera mine, of the State of Chihuahua, was a considerable shipper of mixed sulphide ore to Pueblo, Colo. Arizona and Nevada both figured as small producers in 1905. The ore of Magdalena, N. M., was shipped chiefly to Missouri, Kansas, and Wisconsin, for the manufacture of zinc oxide. Other Western ores are shipped to Mineral Point, Wis., for the manufacture of zinc oxide.

The statistics of ore production by States are given in the following

table. These statistics are based on the production of zinc ore in marketable form, from the standpoint of the zinc smelter. A certain quantity of low-grade ore, treated at Cañon City, Colo., for the manufacture of zinc-lead pigment, is enumerated separately.

PRODUCTION OF ZINC ORE IN THE UNITED STATES.

State.	1904.		1905.	
	Tons.	(g) Value.	Tons.	(g) Value.
Arkansas	(e) 1,900	$66,000	2,200	$96,000
Colorado	(a) 94,000	940,000	105,500	1,529,750
Idaho	*Nil.*	....	1,700	37,400
Kentucky	(d) 958	10,538	(d) 414	6,624
Missouri—Kansas	273,238	9,692,160	258,500	11,455,280
Montana	*Nil.*	....	2,000	25,000
New Mexico	(e) 21,000	168,000	17,800	222,500
New Jersey	(d)280,029	560,058	361,829	723,658
Utah	*Nil.*	....	9,265	120,445
Wisconsin	(c) 19,300	598,300	32,690	1,307,600
Others	(a) 2,600	36,400	(e) 3,800	72,200
Totals	693,025	12,071,456	795,698	15,596,457

(a) Estimated. (b) Production of Joplin district, plus output of southeastern Missouri, the latter as reported by the State mine inspector. (c) According to H. F. Bain, "Contributions to Economic Geology," 1904. (d) Report of State Geologist; crude ore. (e) Partly estimated. (f) Arizona, Nevada, Illinois, Iowa, Tennessee and Virginia. (g) Values are estimated, no direct reports having been received; they are reckoned, in all cases f.o.b. mines, on the basis of the average price for spelter in each year and the average grade of the ore.

IMPORTS OF ZINC ORE INTO THE UNITED STATES.

Source.	1904.	1905.
British Columbia	2,100	8,561
Mexico	?	(a) 32,164
Totals	?	40,725

(a) The actual tonnage of ore imported was somewhat greater than this figure, but it included some mixed ore, which for statistical purposes has been reduced to the zinc ore equivalent.

There was a small importation of zinc ore from Mexico in 1904, the business with that country having been inaugurated in that year, but statistics concerning it are unavailable.

The total supply of zinc ore in 1905, so far as can be enumerated, was 795,698 tons from domestic sources and 40,725 from foreign, a total of 836,423 tons. The situation is clarified if the production of New Jersey be deducted, leaving 474,594 tons, to be compared with the production of 190,294 tons of Western spelter. The Rocky Mountain ore used for the manufacture of spelter amounted to 160,000 tons.

The total—795,698 tons—understates the actual production of ore to some extent, certain ore consumed for the manufacture of oxide being omitted. The deficiency is chiefly in the presentation of the outputs of Wisconsin and "Other States." The aggregate production of marketable zinc ore in the United States in 1905 was doubtless in excess of 800,000 tons.

The United States Smelting Company, at Cañon City, Colo., treated 33,000 tons of zinc-lead ore, averaging 22.7 per cent. zinc, and 8.8 per cent. lead, all of which, except 800 tons from Arizona, was obtained from Colo-

rado. This ore, used for making zinc-lead pigment, has not been included in the above statements.

The exportation of zinc ore from the United States in 1905 was 30,448 tons against 35,333 tons in 1904. This was chiefly New Jersey willemite. There was also exported in 1905 zinc dross to the amount of 5318 tons. This is galvanizers' waste.

Production of Spelter.

The total production of spelter in 1905 was 201,748 short tons. The following table shows the distribution of the production according to States. Of the total, 190,294 tons was Western spelter; the remainder was Eastern spelter, being largely the high-grade brands produced by the New Jersey Zinc Company at Bethlehem and Palmerton, Penn., and by the Bertha Mineral Company, at Pulaski, Va. These companies produce, however, second and third grades of metal, as well as the first grade. The Western metal, also, is produced in different grades, although their values do not cover so wide a range as in the case of the Eastern spelter. "Glendale refined," which ordinarily commands a premium of about 1¼c. per lb. over ordinary prime Western, is produced by the Edgar Zinc Company from selected ores. This is used largely in the manufacture of brass. The remainder of the Western spelter is marketed as "specials," and ordinary "prime Western." The specials are the first of the three draws made daily from the furnace, and this being the spelter distilled at the lowest temperature, is lower in lead and higher in cadmium than the second and third draws. The special brands ordinarily command a premium of 10 to 15c. per 100 lb. over ordinary prime Western.

In the production of prime Western spelter, a few of the smelters continue to use nothing but Joplin ore, but the majority use ore from west of the Rocky Mountains in connection with the Joplin ore, and several of them use it exclusively. The ores from the far West when properly smelted furnish a good grade of spelter, which meets the requirements of galvanizers and the manufacturers of some grades of brass.

There was a large increase in the smelting capacity in 1905, the Caney Zinc Company, of Caney, the Chanute Zinc Company, of Chanute, and the Cockerill Zinc Company, of Altoona, Kansas, and the Grasselli Chemical Company, of Clarksburg, W. Va., whose plants were constructed in 1904, having a full year of operation in 1905. The Caney Zinc Company and Cockerill Zinc Company each added two new blocks of furnaces (608 retorts per block) to their plants in 1905. No other additions to plants were reported, except that the Granby Mining and Smelting Company had under construction one new block (of 320 retorts) at its works at Neodesha, Kansas, which was to be completed about Jan. 1, 1906. Several new plants are, however, now under construction, the most important being those of

Hegeler Bros., at Danville, Ill., which is to comprise 1700 retorts, and the large plants which the Mineral Point Zinc Company is erecting at Depue, Ill., which will have 4800 retorts. Both these plants are to make sulphuric acid, as well as spelter. The Depue works will be equipped with Neureuther-Siemens regenerative furnaces, similar to those employed at Peru, Ill. The Danville plant will employ a modification of the Hegeler furnace. The Danville plant should come into operation before the end of 1906. It will probably be toward the end of the year before the works at Depue are completed. The New Jersey Zinc Company is also making a large extension to its plants at Palmerton, Penn., the number of furnaces being increased from four to twelve. Upon completion of the addition these works will have two Siemens furnaces, and ten Convers & DeSaulles furnaces, the latter having 200 retorts per furnace.

Several new works are now contemplated, but it is not likely that any of them will come into operation during 1906. It is reported that the United States Zinc company will build a new plant at Pueblo, Colo., of the same type and capacity as its present plant, which has six Overpelt furnaces (1440 large retorts). At least one plant is under consideration for erection in Illinois. The Northern coalfield of Illinois now seems destined to be the great center of Western zinc smelting, it being close to the Wisconsin mines, which are rapidly increasing in production, able to command ore from the Joplin and the far West on good terms, and at no great advantage in fuel, as compared with Kansas, since the cost of gas in the latter State has increased materially. However, it appears probable that the manufacture of sulphuric acid as a by-product may be temporarily overdone, although it is the general experience that when a supply of that important commodity is offered, a demand for it rapidly develops.

The Wenona Zinc Manufacturing Company, of Wenona, Ill., which owns a small plant of direct-fired Belgian furnaces, discontinued operation shortly after the middle of 1905. The Cockerill Zinc Company lost two months' operation of two blocks through fire at its Altoona plant. The extension in the operations of the Cockerill Zinc Company was a feature of 1905. This company now operates the smelters at Altoona, Kan., La Harpe, Kan. (formerly La Harpe Smelting Company), Gas, Kan. (formerly Cherokee-Lanyon Spelter Company), Pittsburg, Kan., and at Rich Hill and Nevada, Mo. The resumption of operations by this company at one of the old coal smelters at Pittsburg, Kan., was a noteworthy feature of the year.

The aggregate capacity of the Western smelting works is now very large, but it is to be remarked that the treatment of the large quantity of comparatively low-grade Rocky Mountain ore requires considerably larger furnace capacity than the high-grade Joplin ore.

I estimate that out of the 190,294 tons of Western spelter produced in

1905 about 124,000 tons was derived from ore mined in the Joplin district, about 12,294 tons from ore mined in Wisconsin, Kentucky, southeastern Missouri and Arkansas, and about 54,000 tons from ore mined west of the Rocky Mountains, including British Columbia and Mexico. The quantity of spelter originating in the far West certainly shows a remarkable growth for an industry that is only five years old. In 1904, about 128,000 tons of spelter was derived from Joplin ore. The total production of Western spelter having largely increased in 1905, the relative position of Joplin was materially reduced.

It must be explained why the output of 54,000 tons of Rocky Mountain spelter, which would correspond to only about 162,000 tons of crude ore, does not agree with the statistics previously given by States. This is chiefly because a good deal of that ore was consumed for the manufacture of zinc oxide. The figure—162,000 tons—agrees satisfactorily, however, with that previously stated—160,000—as the known consumption of this class of ore for spelter-making.

The total production of zinc oxide in 1905 was 65,403 tons, which represents 52,322 tons of spelter. Part of this was derived from New Jersey ore, part from Wisconsin, Kentucky and other ore, and part from Rocky Mountain ore. The producers of zinc oxide are few in number, and further analysis of this industry would be impossible without betraying individual interests.

PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1899.	1900.	1901.	1902.	1903.	1904.	1905.
Colorado	....	....	....	....	877	4,906	6,599
Illinois (a)	49,290	37,558	44,896	49,672	49,526	47,607	45,357
Kansas	55,872	57,276	74,270	87,321	87,406	103,721	114,948
Missouri	15,710	20,138	13,083	10,548	9,894	12,056	11,800
South and East (b)	8,803	8,259	8,603	10,698	10,799	13,513	23,044
Total tons of 2000 lb.	129,675	123,321	140,822	158,237	158,502	181,803	201,748
Total tons of 2240 lb.	115,781	110,028	125,734	141,283	141,520	162,324	180,132
Total metric tons	117,644	111,794	127,751	143,552	143,792	164,921	183,014

(a) Up to 1903, inclusive, includes also the production of Indiana. (b) New Jersey, Pennsylvania and Virginia, and (since 1903) West Virginia.

EXPORTS OF DOMESTIC SPELTER FROM THE UNITED STATES. (a)

Year.	Plates, Sheets, Pigs and Bars.		Wares.	Total Value.
	Short Tons.	Value.	Value.	
1896	10,150	$1,013,620	$51,001	$1,112,029
1897	14,245	1,356,538	71,021	1,743,049
1898	10,499	1,033,959	138,165	1,724,188
1899	6,755	742,521	143,232	1,978,295
1900	22,411	2,217,963	99,288	2,317,251
1901	3,390	228,906	82,046	310,952
1902	3,237	300,557	114,197	414,754
1903	1,521	163,379	71,354	234,733
1904	10,073	1,094,490	117,957	1,212,447
1905	5,516	682,254	159,995	842,249

(a) There is a comparatively insignificant re-export of foreign-made spelter and zinc wares.

There was also a production of zinc-lead pigment to the amount of 7200 tons, which required the smelting of 33,000 tons of ore, as previously noted.

IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES.
(In pounds.)

Year.	Sheets, Blocks, Pigs and Old.		Manufactures	Total Value.	Oxide.	
					Dry.	In Oil.
1896	856,044	$25,904	$15,728	$41,632	4,572,781	311,023
1897	2,557,341	95,883	19,431	115,314	5,564,753	502,357
1898	2,742,357	109,624	13,448	123,072	3,342,235	27,050
1899	2,985,463	151,956	14,800	166,756	3,012,709	41,699
1900	2,013,196	97,772	36,836	134,608	2,618,808	38,706
1901	775,881	30,920	42,643	73,563	3,199,778	128,198
1902	1,238,091	46,713	37,191	83,904	3,271,385	163,081
1903	728,614	30,900	18,938	49,838	3,487,042	166,034
1904	933,474	44,326	11,918	56,244	2,585,661	224,244
1905	1,042,081	51,052	12,390	63,442	3,436,367	342,944

EXPORTS OF ZINC ORE AND ZINC OXIDE FROM THE UNITED STATES.

Year.	Ore.			Oxide.		
	Short tons.	Value.	Value per ton.	Short tons.	Value.	Value per ton.
1896	(a)2,324	$47,408	$20.40	(b)	....	....
1897	9,251	211,350	22.85	1,859	$104,140	$56.02
1898	11,782	299,970	25.50	3,925	252,194	64.25
1899	28,221	725,944	25.90	5,343	366,598	68.61
1900	42,062	1,134,663	26.98	5,656	496,380	87.76
1901	44,146	1,167,684	26.45	4,561	393,259	86.22
1902	55,733	1,449,104	26.00	5,358	433,722	80.93
1903	39,411	987,000	25.04	7,215	578,215	80.14
1904	35,911	905,782	25.22	8,157	628,494	77.05
1905	30,946	848,451	27.41	11,280	810,203	71.83

(a) Includes oxide. (b) Included in ore.

CONSUMPTION OF SPELTER.

The stock of spelter in the hands of smelters at the beginning of 1905 amounted to about 6500 tons. The production during the year was 201,748 tons. The imports amounted to 521 tons. The total supply was consequently 208,769 tons. The exports of spelter during the year were 9515 tons. The stocks in the hands of smelters at the end of the year amounted to 4000 tons. The domestic consumption was consequently 199,254 tons.

I have made an attempt to distribute this consumption according to use. So far as I am aware, this has not previously been attempted in a systematic manner, and there are but few data with which to make comparison. In 1892 it was estimated by W. H. Seamon that out of a total consumption of 78,040 tons, the galvanizing trade used 35,000; the brass-makers, 20,500; the rollers of sheet zinc, 15,500; the desilverizers of lead bullion, 3500; while the remaining 3540 tons were employed for miscellaneous purposes. In 1898, according to statistics which I had occasion to collect, out of a total consumption of 105,000 tons, the galvanizers used about 55,000 tons (52 per cent.); the brass-makers, 24,000 tons (23 per cent.);

the rollers of sheet zinc, 20,000 tons (19 per cent.); the lead desilverizers, 1500 tons (1.5 per cent.); and miscellaneous consumers, 4500 tons (4.5 per cent.). In THE MINERAL INDUSTRY, Vol. VIII, it was estimated that, of the consumption of spelter in the United States in 1899, about 50 per cent. was used in galvanizing, 15 per cent. in brass-making, 20 per cent. in the form of sheets, and 15 per cent. for other purposes. The agreement between those figures and my own for the previous year is very close with respect to the galvanizing and sheet zinc industries. With respect to brass and consumption for other purposes, I am disposed to consider that THE MINERAL INDUSTRY underestimated the former and overestimated the latter.

For 1905 I received direct reports from consumers accounting for 163,562 tons of the consumption during the year. Twenty-one galvanizers reported the consumption of 92,766 tons. Seventeen brass-makers reported the consumption of 32,888 tons. The rollers of sheet zinc, of whom there are six, made reports with two exceptions, enabling the estimate of 34,000 tons as the consumption for this purpose to be made with close approximation to the truth. The consumption of spelter for the desilverization of lead is estimated on the basis of the desilverized lead produced, a consumption of 0.8 per cent. of spelter being reckoned.

The statistical reports are undoubtedly most incomplete with respect to brass and the consumption of zinc for other purposes than those enumerated above. There are a large number of small brass-makers in the United States who consume from 5 to 250 tons of spelter per annum, many foundries being conducted in connection with other manufacturing enterprises. Among the consumptions of zinc for other purposes are the use of the metal in making castings, in connection with which a good deal of high-grade spelter is employed, and also for such purposes as the manufacture of battery zincs, for which virgin spelter is remelted and molded. Spelter is also employed for the manufacture of zinc chloride, zinc sulphate, and other chemical products. For all these purposes, however, a good deal of scrap from the cutting of sheet zinc is utilized, even certain galvanizers being purchasers of that material.

My reports indicate that the consumption of spelter in 1905 may be classified approximately as follows: Galvanizing, 50 per cent.; brass-making, 26 per cent.; sheet zinc rolling, 17 per cent.; lead desilverizing, 1¼ per cent.; other purposes, 5¾ per cent.

Incidentally this gives a rough idea of the consumption of copper for brass-making. If 52,000 tons of spelter were used for that purpose, there must have been employed at least 104,000 tons of copper for the same purpose. This figure corresponds to about 34 per cent. of the domestic consumption of new copper in 1905.

The great center of the American brass industry is the Naugatuck valley,

Connecticut; of the galvanizing industry, the center is western Pennsylvania and eastern Ohio, within the districts commanded by Pittsburgh, Penn., Wheeling, W. Va., and Youngstown, Ohio. A large proportion of the spelter produced in the United States is consumed in those districts.

Mining Conditions.

The zinc industry of the United States in 1905 was characterized by a large production, a high range of prices for spelter and a shortage in ore supply, which caused the market for raw material to hold at a very high level, but left only a small margin to the smelter, notwithstanding the strong market for the metal. The year was, therefore, extremely prosperous to the producers of ore, but not so prosperous to the smelters. Some of the latter were, indeed, obliged at certain times to put a portion of their furnaces under dead fire.

The production of the Joplin district was a little smaller than in 1904. The new openings of ore are not compensating for the exhaustion of the older mines. A large part of the Joplin output at the present time is derived from the "sheet ground." Some excellent mines are being worked in this "sheet ground," and at least one that is remarkable, but on the whole this class of ore deposit is rather lean and the cost of mineral obtained from it is high. It appears to be recognized that this district is now affording as large an output of ore as can reasonably be expected from it, and further supplies must be looked for in other quarters. Indeed, if it had not been for the prominent position which the zinc ore of Colorado and other Western States has assumed in the market, there is no doubt that disaster would have overtaken the domestic zinc industry. The delivery of large supplies of ore from the Rocky Mountains, however, has enabled several smelters of Kansas to withdraw from the Joplin district. Their ability to obtain the comparatively low-grade ores from points remote in the Rocky Mountains is due to the very low rates on such traffic which have been made by the railways. If the freight rates were reduced to the ton-mile basis they would figure out perhaps as low as on any commodity traffic on this continent.

The extensive importation of zinc ore of British Columbia, and especially from Mexico, led to a dispute between the domestic producers of ore on the one hand and the smelters who were making the importations on the other hand as to the proper construction of the Dingley act with respect to such importations. There was never any doubt as to the importation of calamine, since that ore is explicitly put on the free list in the Dingley tariff. There was a question, however, as to whether blende should be assessed at the rate of 20 per cent. *ad valorem* under the clause in the tariff which establishes that rate on ores and minerals unenumerated, or whether it should come in free under another paragraph

of the act. It was finally decided by the Treasury Department, Feb. 10, 1906, that the term "calamine" in the Dingley act referred only to the hydrous silicate of zinc, and that all other ores of zinc shall pay the duty of 20 per cent. This classification of calamine is contrary to the mining, metallurgical and commercial practice of a century, and the case is to be taken to the courts.

An important feature of the year was the increased extent to which some of the zinc smelting companies invested in mining property. Several of these companies have had agents in the field for two years or more in the search for desirable zinc mines, and during 1905 several purchases of such property were made. This new policy is directed toward the assurance of ore supply under the control of the smelting company itself. This will tend to give the industry more stability.

East of the Mississippi there was a good deal of activity in the Wisconsin field, where the production materially increased. The work done in that field, however, was largely of a preliminary character, and ought to show the results in 1906 and following years, rather than in the last year. It has been found that the separation of the blende and the marcasite, which is the common ore of the district, can be economically and efficiently accomplished magnetically and otherwise. Various plants have been erected for that purpose, and more may be expected to follow. Some of the large zinc producing companies made extensive investments in the Wisconsin field, and there is a general confidence that this will prove to be an important source of zinc. Geographically it is more advantageously situated than any of the zinc producing districts west of the Alleghenies.

Some developments were made in the Holston River district of Tennessee, with encouraging results, it is claimed. The deep mining which has been inaugurated in that district is a new feature. The output of Tennessee in 1905, however, was insignificant. In New Jersey the development of the great mine at Franklin was continued on the same lines as in previous years, and it made the usual large output.

Zinc Mining in Tennessee.

Active operations in the way of deep mining were conducted in Tennessee in 1905. There are three distinct fields, namely, the Lead Mine Bend, the Straight Creek and the Holston River. In the Straight Creek field, the Tennessee Zinc Company, of Cincinnati, O., is mining ore carrying 15 per cent. zinc and 20 per cent. lead; this company recently contracted for a 100-ton concentrating plant.

The greatest development work has been done in the Holston River field; the Roseberry Zinc Company has been prospecting with a calyx drill, and has demonstrated that it has a deposit of good milling ore

at a depth of 158 ft., and extending to 202 ft., which is the extreme depth drilled. It now has a shaft down 172 ft., and is drifting off on a 14-ft. face. It has produced experimentally a concentrate running 61 per cent. car load lot, with a very satisfactory percentage of extraction.

The Holston Zinc Company (which joins the Roseberry property) has also done a large amount of drilling, proving that the Roseberry ore shoot extends through its property; this company has sunk a shaft to a depth of 200 ft., and has built a concentrating plant, which should be in operation in 1906.

The Holston river field extends from a point four miles east of Knoxville for over 20 miles, paralleling the Holston river and the Southern railway. Along this entire field considerable prospecting has been done, showing that the formation of dolomite carrying ore is continuous; the blende is entirely free from lead and carries less than 1 per cent. iron, with no other impurities. Shipping facilities are of the best; fuel and labor are cheap. It appears likely that this district may figure as a zinc producer of some consequence in the near future.

Missouri and Kansas.

By Jesse A. Zook.

Notwithstanding the falling off in the tonnage of ore produced, 1905 was a boom year in the Joplin district. Zinc concentrate sold $55 per ton the first week, advancing to $58 the next week. The next two weeks recorded the highest price ever paid for zinc ore, namely, $60 per ton. In the first week of February, $2 was taken off; in the second week, an additional $1.50; during the last half of that month, $56 was paid. March began with zinc at $54.50, dropping to $45 at the end of the month; through April there was a steady advance to $48 at the close, and to $48.50 for the first week of May. Then the pendulum again swung backward, continuing until it reached $43.50 in the third week in June. From this, the lowest point of the year, prices advanced $1 to $3 per week, until the highest price came in the second week in August, $59 per ton, and within $1 of the January record-breaking price. During the succeeding three weeks the price declined $10 per ton, regaining an upward tendency the second week in September, and continuing upward to $55 in the first week of October, then down two weeks, touching $51.50, and returning to $55 in another two weeks, where it remained throughout November. During December the price fluctuated between $53 and $55 per ton.

The highest price paid for first-grade zinc concentrate was $60 per ton in January; the lowest price for this grade was $43.50 in June. The average price for all grades of sulphide and silicate ores during the year was $44.88 per ton.

The total zinc shipment for 1905 was 252,435 tons, at a value of $11,-335,280. The total lead shipment for 1905 was 31,679 tons, at a value of $1,968,480. The total shipment of both ores was 284,114 tons, at a total value of $13,302,800.

The following table gives the shipments of zinc and lead ore for the last 12 years:

Year.	Zinc tons.	Lead Tons.
1894	147,310	32,190
1895	144,487	31,294
1896	155,333	27,721
1897	177,976	30,105
1898	234,455	26,687
1899	255,088	23,888
1900	248,446	29,132
1901	258,306	35,177
1902	262,545	31,625
1903	234,873	28,656
1904	267,240	34,362
1905	252,435	31,679

The smaller output in 1905 was directly attributable to an extraordinary rainfall, amounting to 50 in. In January and February occurred the coldest weather known in the district since the discovery of ore; during that time there occurred a succession of snow storms and cold rain storms covering the district with a mantle of ice 6 to 8 in. in thickness for a period of six weeks. What ore was mined during this period was frozen solid in the bins, and for shipment had to be blasted out. The spring was rainy and cold, and the mines had but fairly recovered from an unusual winter and spring when July floods sent an inflow of water into them. August followed with heavy rains. After a comparatively dry September, October brought additional heavy rains and came near inundating many mines again. For the first time in the year conditions assumed a normal basis in the latter part of October; after this, conditions remained favorable until labor troubles at the end of November upset the prospects again. The United States Supreme Court handed down a decision during the month sustaining the constitutionality of the eight-hour labor law, enacted by the State legislative session of 1900–1, making a day for miners eight hours. It developed, during the controversy arising over the adjustment of this law to the local situation, that the last legislative session had enacted a further law embracing all workmen about a mine under the eight-hour system. Underground men had been working eight hours, mill men 10 hours and engineers 12 hours. Owing to high ore prices and no advance in the general wage scale, employees began demanding full pay for the eight-hours' work. Adjustments were made between the operators and employees, but analysis of the Supreme Court ruling on the eight-hour law made it appear clear that the later legislative action, embracing top workmen under the eight-hour law, would be held uncon-

stitutional, leaving the effective eight-hour law applicable only to men working underground. Labor agitators, however, caused the arrest of some operators in January, and it is possible the point at issue will be carried through the courts.

The introduction of electrical energy from a central water-power station, and the piping of natural gas to the mines of the western part of the district during 1905, inaugurated an era of improvement and economy in mining. It is producing a pronounced evolution in the mine and mill motive-power equipment of the district. In the construction of new mills during the latter half of 1905, electric motors or gas engines have been installed, or provision made for them. At some mines boilers and steam-engines have been removed. At other mines, gas is piped under the boilers for generating steam. Supplied at 10c. per 1000 cu. ft., even thus extravagantly used, it is proving an economical fuel, compared with Kansas bituminous coal. Another year should show a marked reduction in cost per ton of concentrate, unless the demands of labor absorb the savings that can be made.

A new method of mine and mill operation has been introduced at the Golden Rod mine of the Underwriters Land and Mining Company. Here two inclined shafts have been driven in opposite directions, on an angle of 45 deg., to the horizontal or "blanket" orebody. Metal conveyors raise the ore in the place of the ordinary direct-lift rope and bucket plan of the district. The milling and concentrating apparatus are improved to the fullest extent, along the lines of the successful mills of the dstrict, in sections giving a combined capacity of 1000 to 1400 tons of zinc-lead concentrates per week. This capacity is five times greater than that of any other mill, and ten times greater than that of the ordinary mill of the district.

Further effort is being made to develop the lower ore stratum, indicated by drill at Aurora. One shaft was sunk to the 325-ft. level two years ago, but this lower level was abandoned last year on account of water. Another company is now sinking a shaft to this lower level.

AVERAGE MONTHLY PRICE OF ZINC BLENDE ORE AT JOPLIN, MO.
(Dollars per short ton.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1900.....	30.23	29.36	28.45	28.42	26.92	25.00	24.23	25.67	24.25	24.25	24.45	25.40	26.50
1901.....	23.72	23.96	23.70	24.58	24,38	24.22	24.68	23.88	21.63	21.63	26.15	28.24	24.21
1902.....	26.75	27.00	28.00	28.85	29.23	34.10	34.37	32.50	33.58	33.58	32.10	29.25	30.73
1903.....	31.50	32.50	35.75	37.75	36.60	36.50	36.00	36.00	34.40	34.40	30.75	30.00	34.44
1904.....	32.12	34.00	36.00	36.40	34.63	32.62	35.00	37.00	40.40	40.00	44.25	46.13	37.40
1905.....	51.94	53.65	47.40	43.93	43.74	40.75	43.00	50.24	46.80	49.37	50.37	47.67	47.40

Zinc Mining in Wisconsin.

By E. W. Moore.

After nearly 50 years of mining, more or less active, the Platteville zinc and lead district, as the southwest Wisconsin zinc and lead fields

are locally called, is enjoying a period of prosperity. Today there are 20 to 30 large dividend-paying mines and many smaller ones, as against only two or three a little over a year ago. The chief causes for this improvement have been the high prices for ore, and the perfection of the system of magnetic separation.

Five years ago it was generally thought that it was only a question of a short period before the mines would be a thing of the past, as the richer veins continued to carry more and more pyrite, locally called sulphur, which cut the price of zinc ore down below a profitable point. The late Richard Kennedy, one of the best known local zinc men at that time in the district, returning from the Joplin district, was imbued with the idea that if the Joplin operators could make money on 4 and 5 per cent. propositions, the operators in the Platteville district surely could on 15 and 20 and even as high in some instances as 50-per cent. ore, provided proper methods and modern machinery were employed.

The first concentrating plant was built three years ago, on the Oldenburg property, by Mr. Kennedy and his associates. Since that time the Kennedy companies alone have installed 10 different concentrating plants and separators, all of which today are paying dividends. Owing to the fact that the ore buyers heavily penalized the pyritic zinc ores, many of the mines never paid a dollar in dividends until after installing a roasting and magnetic separating plant.

The first plant of this kind was installed at Meeker's Grove camp, on the Trego property. The results were so satisfactory that the builders of the plant were more than pleased, but not altogether satisfied as to values saved. The latest Galena type of roaster, in connection with the Cleveland-Knowles separator, as now generally installed, shows an efficiency of 90 per cent. and produces a product of which the heads assay as high as 65.4 per cent. zinc and 0.9 per cent. iron with only 2.05 per cent. zinc in the magnetic tailings, from an original product assaying 19.2 per cent. zinc and 29.0 per cent. iron. Of course, the above is the result of the best single test, the average result being in the neighborhood of 60 per cent. zinc and 2.9 per cent. iron finished heads, from an average original concentrate assaying 40.2 per cent. zinc and 16 per cent. iron.

Another interesting innovation has been the change from gasoline to steam power and the installation of central electric power plants. One of the most serious drawbacks has been the lack of competent labor, but this difficulty is fast being overcome, through both education and importation.

There are today in active operation 47 concentrating plants and 12 separators, two of which obtain ore merely by buying from producers.

One of the most important additions to the interests of the district was the completion of the Mineral Point Railroad, connecting with the Chicago,

Milwaukee & St. Paul and Chicago & Northwestern, and entering the three principal camps in the northern part of the district—Highland, Linden and Mifflin. The Chicago & Northwestern has built spurs to all the principal mines along its line, as also has the Chicago, Milwaukee & St. Paul.

The Mineral Point Zinc Company during the last 18 months has been buying up all the producing mines it could secure, and by merging operations has eliminated considerable expense. This company, also, has enlarged its oxide works, spending about $500,000, and is at the present time adding quite extensively to its acid plant. It also installed a battery of Galena type roasters and Cleveland-Knowles separators.

The demand for zinc ore, as well as lead, exceeded the supply. At the end of the year there was not a pound of surplus ore in the bins. The Platteville district was almost wholly developed by home money. More attention has been paid of late by outside capital, but the majority are men who have had experience in the West and Southwest, and who are being attracted by the comparatively small outlay necessary to prove up a property. Recently an interurban electric freight and passenger line has been proposed and the right of way is now being surveyed. This line will give shipping facilities to a great number of the mines that have been depending altogether on team and wagon, the hauls running from 5 to 15 miles. The foundries at Galena and Platteville have increased their capacity, to take care of the increased business, which is quite an item to the operator. More improved methods and modern machinery were introduced into the district last year than ever before.

Virginia Lead and Zinc Mining.

By J. A. Van Mater.

A strong metal market tended to stimulate prospecting; old locations are being re-examined and some new ones investigated; nothing new, however, of importance has yet resulted from this work. The two most promising properties are that of the Albermarle Zinc and Lead Company at Fabers, Nelson county, and the Cedar Springs Zinc Mine and Development Company, near Rural Retreat in Wythe county; at both of these places mills have been equipped to concentrate mineral.

At the former mine, the blende and galena occur, with fluorspar and quartz, in mica schist. Air jigs are used for the separation, but no mineral has been shipped. At the Cedar Springs property, a shaft has been sunk some 50 ft., and some other development work has been done, which encourages the owners to believe that they are warranted in erecting a mill. The ore should prove an easy milling proposition, if it occurs in sufficient quantity, as the blende is distributed through the brecciated rock in fairly coarse form; but, even if the property opens up well, it will be some time before it can be counted a producer. There has likewise

been some prospecting for lead in the Rye Valley district in Smyth county; but no deposits of importance have been disclosed.

The only zinc and lead produced in this State during 1905 was by the Bertha Mineral Company, from the old Wythe lead and zinc mines at Austinville in Wythe county. The mines have been worked, more or less continuously, since about 1750 (when they were opened for lead); but it was not until about 20 years ago that any attention was paid to the zinc, which occurs as silicate and carbonate, near the surface between chimneys of limestone. In the early days, when lead was the only mineral sought, large quantities of these surface ores were wasted; but in later years they have been carefully saved. This property, next to the Bertha mines, has been the most important producer in the State, and is now the only one shipping any zinc ore. The output in 1905 was 651 tons of zinc concentrate, and 89 tons of lead concentrate; besides 348 tons of low grade zinc oxide, recovered from the mill tailing on the Bertha property, this oxide being turned into spelter. The output was much curtailed by the burning of the oxide plant in June, and by the breaking away of part of the dam in the river, which necessitated extensive repairs, entailing a shut down of all power for four months. While the output of soft ore from this property is likely to be small in the future, there has been considerable development work done in the rock underlying the soft-ore deposit, in the hopes that sulphide ores of sufficient importance would be found to make a new mine of this property. This work has been vigorously prosecuted throughout the year, and has shown results which on the whole are encouraging; but it will take considerable more work before the future of this mine can be determined.

The Bertha mines (likewise owned by the Bertha Mineral Company), which were abandoned about eight years ago as a zinc property, have been producing limonite iron ore, which had overlaid the deposits of soft zinc ore. There was a hope that, after the stripping of a considerable portion by the mining of the iron ores, other deposits of zinc ore would be uncovered; but such has not proved to be the case, with one exception, where a small basin was encountered, but of no particular importance.

Zinc Production of the World.

Complete official statistics of the production of spelter are not yet available. According to the statistics of Henry R. Merton & Co., Ltd., the production in Europe and America in 1905 was 647,585 long tons, against 615,290 in 1904 and 562,325 in 1903. This represents practically the total spelter production of the world, the only omissions being the outputs of Sweden and Norway, Saxony and Australia, which in the aggregate are comparatively insignificant. The total zinc production of the world is, however, about 7.5 per cent. larger than the spelter pro-

duction, because of the important output of zinc oxide in the United States, which is produced directly from ore. The statistics of Messrs. Merton in detail are as follows:

Country.	1903.	1904.	1905.
Belgium	129,000	137,780	143,165
Rhine District	61,315	64,360	66,185
Holland	11,515	12,895	13,550
Great Britain	43,415	45,490	50,125
France and Spain	41,780	48,310	49,575
Silesia	116,835	123,695	127,895
Austria and Italy	9,025	9,100	9,210
Poland	9,745	10,440	7,520
United States	139,695	163,220	180,360
Total	562,325	615,290	647,585

The great increase in the spelter production of the United States in 1905 puts this country very near to the leading position, which it is likely to take within three or four years. The aggregate output of Silesia and the Rhenish districts in 1905 was 194,080 long tons, and this is practically equivalent to the output of Germany entire, the produce of the works at Freiberg, Saxony (which are not included), being only a few hundred tons. As a producer of zinc the United States has been in the lead for several years, because of its large production of zinc oxide. In 1905, this amounted to 65,403 short tons, equivalent to 52,322 short tons (46,716 long tons) of zinc. The total production of zinc in the United States in 1905 was, therefore, considerably upward of 250,000 short tons.

The production credited to "France and Spain" in the Merton statistics is mostly due to the former, there being only one smelter in Spain. Similarly, there is only one smelter in Italy, and its output being very small, the figures credited to "Austria and Italy" are due chiefly to the former.

Every country showed an increase in 1905 except Poland. The falling off in Poland, amounting to upward of 25 per cent., was due to labor troubles. The largest proportional increase was made by the United States, which gained 10.5 per cent. Great Britain was, however, only a trifle behind, its gain having been 10.2 per cent. The progress in spelter production in Great Britain during the last few years has been particularly noteworthy. In 1901, its output was only 30,055 long tons, or six per cent. of the world's output; in 1905 it was nearly eight per cent. of the world's total. The minor position occupied by the British smelters previous to 1901, and even at the present time, was and is due to their own backwardness. Great Britain has much greater resources of zinc ore than some of the Continental countries which are important in the smelting business—Holland, for example, which has no ore at all; and Belgium, which has only a little left of its once grand resources. These countries are depend-

ent upon foreign mines for their ore supply. Great Britain, with its abundant supply of fuel, admirable seaport locations, and important ocean traffic, is in an equally good position, save for some disadvantage with respect to wages for labor.

We may confidently expect that before long Great Britain will acquire a much more important position in the zinc smelting industry. There is now on foot a movement to utilize more extensively the vast zinc resources of Broken Hill, where there is literally "in sight" and developed a greater tonnage of zinc ore than few, if any, other districts can show. Three strong companies, owning those vast resources of ore, are making plans to undertake the smelting of it, and in this connection the establishment of smelteries in Great Britain is projected. Developments on this line will be awaited with interest. In the meanwhile increased attention is being devoted to zinc mining in Great Britain, and there was in 1905 more activity in that direction than for a long time.

PRODUCTION OF ZINC IN THE WORLD. (*a*)

(In metric tons.)

Year.	Austria.	Belgium.	France.	Germany	Holland.	Italy.	Russia.	Spain.	United Kingdom	United States.	Totals.
1896......	6,888	113,361	45,585	153,082	4,770	*Nil.*	6,257	6,133	25,278	70,432	421,786
1897......	6,236	116,067	38,067	150,739	6,600	250	5,868	6,244	23,805	91,070	444,964
1898......	7,302	119,067	37,155	154,867	6,700	250	5,664	6,031	28,387	103,514	468,937
1899......	7,192	122,843	39,274	153,155	6,235	251	6,331	6,184	32.322	117,644	491,331
1900......	6,742	119,315	36,305	155,799	6,845	547	5,963	5,611	30,207	111,794	465,438
1901......	7,558	127,170	37,600	166,283	7,855	511	6,090	5,354	29,877	127,751	516,049
1902......	8,309	124,780	36,282	174,927	9,910	485	8,280	5,569	40,244	143,552	552,338
1903......	8.949	131,740	27,462	182,548	11,515	126	9,901	5,134	44,110	143,792	560,017
1904......	9,159	137,323	43.196	193,058	12,895	189	10,607	5,887	46,218	164,921	623,453
1905......	9,210	143,165	44,075	198,208	13,550	(*b*)	7,520	5,500	50,125	183,014	654,367

(*a*) From the official statistics of the various Governments, except 1905, for which year the figures reported by Henry R. Merton & Co. have been used where the official statistics were unavailable. (*b*) Included in Austria.

The high price for spelter in Europe in 1905 is understood to be due to deficiency in the smelting capacity, in comparison with the increase in the industrial demands, rather than to any shortage of ore. Large supplies of ore of inferior character have been obtainable from Australia. Indeed, the supply of ore at one time was so large as to be unsalable. That continent appears to be destined to take an important position in the zinc industry of the world. There is no other place where there is so much zinc ore literally in sight as in the great tailing piles at Broken Hill. The mines continue to show large reserves of mixed sulphide ore, and every year make large additions to the accumulations on the surface. Although the separation of this ore is a difficult problem, the magnetic separation and flotation processes have succeeded in making out of it a marketable grade of zinc ore. During the latter portion of 1905 a large number of the great dumps at Broken Hill were purchased by the Zinc Corporation, Ltd., a new British company, which is planning to

go into the zinc business on a very large scale. The Sulphide Corporation is now operating a small smeltery at Cockle Creek, N. S. W., and the Broken Hill Proprietary Company is building a plant at Port Pirie.

On the subject of zinc production in Europe in 1905 Rudolf Wolff, of Kreuger & Co., wrote as follows in their annual review:

"Zinc ore throughout the year was offered in tremendous quantities, and from all quarters; interest was chiefly directed to the huge accumulation of tailings on the Barrier Reef of Australia; contracts have been entered into for bringing part of these vast supplies to the market. Considering the copiousness of ore supplies during 1905, it will be a matter of surprise that production of spelter did not increase on a greater scale. The reason is that most works found during the year that they were smelting to their fullest capacity, and unless they enlarged their plants they really could treat no more ore; the building of new furnaces meant a capital outlay which had to be seriously considered in view of a possible break in the high prices, and subsequent unremunerative over-production; moreover more furnaces meant more workmen, and a zinc-distilling workman can be made, but not found; all this takes time, and with the fear of the completion of arrangements happening too late, it is scarcely surprising that many hesitated about increasing their capacity."

However, there was some new construction in Silesia during 1905, as is mentioned in a subsequent article, and a new plant of large capacity is now being erected by the International Metal Company at Hamburg, Germany, while new plants are to be erected at Honfleur (near Havre) and at Montagne, France. Two new plants for the smelting of Australian ore in Wales are under consideration, but neither is likely to be begun in 1906. Anyway, it appears that any deficiency in European smelting capacity will shortly be repaired.

PRODUCTION OF SPELTER BY EUROPEAN COMPANIES.
Reported by Henry R. Merton & Co. (In long tons.)

Great Britain.

Company.	1902	1903	1904	1905
Vivian & Sons	5,915	8,030	8,450	9,475
English Crown Co., Ltd	6,165	5,635	6,725	7,645
Dillwyn & Co	6,520	7,150	7,370	7,425
John Lysaght (Ltd.)	3,100	3,335	3,340	3,695
Swansea Vale Spelter Co	2,160	2,300	2,405	2,695
Villers Spelter Co	1,250	2,500	2,600	2,400
Pascoe-Grenfell & Sons, Ltd	2,040	2,465	2,600	2,790
Various[1]	12,460	12,000	12,000	14,000
Total	39,610	43,415	45,490	50,125

[1]Chiefly from the refining of hard zinc and other by-products.

Holland.

Company.	1902	1903	1904	1905
Société de la Campine	9,910	11,515	12,895	13,550

Belgium.

Company.	1902	1903	1904	1905
Vieille Montagne	49,640	54,745	58,575	62,905
Biache St. Vaast	3,985	3,990	3,895	3,975
G. Dumont & Frères	13,535	13,720	13,265	12,445
Austro-Belge Co.	9,770	10,050	10,040	10,335
Nouvelle Montagne Co.	8,860	8,260	10,095	10,805
Société Prayon	12,515	13,140	15,310	15,535
Société de Boom	5,590	6,020	6,055	6,070
L. de Laminne	6,965	7,120	7,090	7,050
Escombrera Bleyberg Co.	4,860	5,115	5,445	5,450
Cie. d'Overpelt	6,310	6,840	8,010	8,595
Total	122,030	129,000	137,780	143,165
Total production of the Vieille Montagne in Belgium, Germany and France	69,755	76,905	83,580	87,620

Rheinland and Westphalia.

Company.	1902	1903	1904	1905
Stolberg-Westfalen	18,140	20,750	22,365	24,805
Rhein-Nassau	10,015	10,330	10,190	10,700
Grillo	7,760	9,420	10,610	10,740
Vieille Montagne	7,605	8,300	8,545	8,810
Märk-Westf. Berg.-Ver.	7,035	7,070	6,955	6,100
Berzelius	5,135	5,445	5,695	5,030
Total	55,690	61,315	64,360	66,185

France and Spain

Company.	1902	1903	1904	1905
Asturienne	21,165	20,330	22,950	22,615
Vieille Montagne	12,510	13,860	16,460	15,905
Malfidano	3,450	5,090	6,155	7,885
St. Amand	2,415	2,500	2,745	3,170
Total	39,540	41,780	48,310	49,575

Silesia.

Company.	1902	1903	1904	1905
Schlesische Akt-Gesellsch.	27,560	27,445	29,780	29,735
G. von Giesche's Erben	25,530	26,160	26,520	26,230
Fürst Hohenlohe	28,170	28,875	29,660	30,020
Graf. Henckel V. Donnersmarck	18,400	18,440	19,665	20,635
Fürst v. Donnersmarck	7,905	8,135	8,485	8,835
(a) O. S. Zinkhutten, A. G. Kattowitz	6,430	6,620	8,390	9,865
O. S. Eisenindustrie A. G.	1,130	1,310	1,195	1,675
Arar	155	150	....	....
Total	115,280	116,835	123,695	127,895

(a) Formerly O. S. Eisenbahn-Bedarfs A. G. and H. Roth.

Austria.

Company.	1902	1903	1904	1905
Sagor	365	470	670	.....
Cilli	2,965	2,880	2,640	2,680
Siersza und Niedzieliska	2,825	2,755	2,610	2,870
Trzebinia	2,185	2,725	3,040	3,630
Total	8,340	8,830	8,960	9,180

Italy.

Company.	1902	1903	1904	1905
Societa di Monteponi	120	195	140	30

The Zinc Industry in Belgium.

The zinc industry in Belgium is almost entirely concentrated in the province of Liége, but there are a few works in the provinces of Anvers (Antwerp), Namur and Limbourg. The spelter production of Belgium is, after that of Germany, the most important of Europe. The statistics for 1904, which is the last year for which official figures are available, are instructive as to the present condition of the smelting industry, especially in showing to how great an extent the Belgium smelters are dependent upon imported ore.

The spelter production in 1904 amounted to 139,847 metric tons, of which 125,570 was produced in the province of Liége. The Belgium ore consumed in this production amounted to only 3050 tons, which was less than one per cent. of the total consumption; 304,320 tons of ore was imported. The following table shows the sources of ore supply during the last four years, in metric tons:

Country.	1902	1903	1904	1905
Italy and Sardinia	74,740	71,674	66,538	96,696
France	25,522	27,268	32,288	51,870
Sweden and Norway	19,333	24,388	24,867	34,896
Germany	22,365	13,252	12,016	.....
Spain and Portugal	52,993	66,566	74,762	51,537
Algeria and Tunis	26,511	32,595	37,483	58,571
Greece	361	1,486	4,896	14,285
Australia	5,288	1,114	18,274	100,065
England	9,017	9,200	6,447	12,080
America	26,227	31,133	21,806	14,094
Turkey	509	789	495	.....
Japan	.....	.....	.....	6,396
Elsewhere	2,146	4,416	4,448	61,480
Totals	265,012	283,880	304,320	501,970
Belgium	5,750	5,355	3,050	.....
Grand totals	270,762	289,235	307,370	.....

The statistics for 1905 have been taken from a recent consular report, and are subject to correction upon receipt of the official statistics. The entire importation of ore is made through the port of Antwerp, whence the ore is transported to the smelting works by the canals, of which an extensive system ramifies through this part of Belgium. Comparatively cheap transportation is thus enjoyed. In some cases the ore goes directly to the smelting works; in other cases it is interrupted in transit for roasting at sulphuric acid works, the burnt ore being thence forwarded to the smelteries. Thus, all the sulphide ore received by the Vieille Montagne Company is roasted at Baelen-Wezèl on the Campine canal. The manu-

facture of sulphuric acid in Belgium is an important industry, the output in 1903 having amounted to 310,000 metric tons of 60 deg. Beaumé, a large proportion of which was derived from blende roasting.

The sheet-zinc rolling mills in 1904 produced 41,492 tons of material, the average value of which was 592.31 francs per ton, which may be compared with the average value of 552.17 francs per ton for spelter. This shows an average differential of only 40.14 francs, or about $8 per ton, which is quite different from what exists in the United States. Zinc oxide is made at only one works, the output of which in 1904 was 8500 tons, all of which is produced by the combustion of spelter.

The Silesian Zinc Industry in 1905.

By Paul Speir.

The business in 1905 was extraordinarily satisfactory, particularly during the second half of the year when consumers came into the market with large orders. The result was that prices rose to a hight which had previously not been attained since 1899, and before that since 1873. The unsually favorable situation of the market led naturally to plans for the increase in capacity of existing works, and the erection of new works.

The spelter production of Upper Silesia in 1905 was about 128,000 metric tons. The monthly range of the market, for usual brands, per 50 kg., f. o. b. Breslau, was as follows: January: 24.90—25.25—24.75 marks. February: 24.75—24.90—24.53. March: 24.35—23.85. April: 23.85 to 23.75—23.80. May: 23.85—23.50—23.85. June: 23.85 to 24.10—23.90. July: 23.90—24.25—24. August: 24—25.75. September: 25.80—26.50—26—27.50. October: 27.75—28.25. November: 28.25—28—28.25. December: 28.25—28.45.

The capacity of the Bernhardihütte was increased by the erection of new furnaces. At Kunigundehütte a new plant, including a roasting installation and a sulphuric acid works, was erected. The Kunigunde, Klara and Franz works, belonging to H. Roth, together with the Rosamunde works belonging to the Oberschlesischen Eisenbahn-Bedarfs-Aktiengesellschaft, were consolidated and taken over by a new company called the Oberschlesischen Zinkhütten-Aktiengesellschaft. The entire mining and smelting business of Prince Hohenlohe was taken over by a company called the Hohenlohe-Werke-Aktiengesellschaft.

In view of the steadily increasing use of blende by the smelters, which results in an enrichment in the zinc content of the charge for their furnaces, the smelters of Silesia are more and more changing their furnaces from the old type to the Rhenish type. Further progress in this direction was made in 1905. At the end of the year there were in use 316 furnaces, with about 12,500 muffles of the old type, with a single row of muffles; and

162 furnaces, with ahout 15,000 muffles, of the Rhenish type, with several rows of muffles.

In the present practice of zinc smelting in Silesia there is a very serious loss of metal, this amounting to approximately 20 per cent. During the very recent years, however, this loss has been considerably reduced at some works in Silesia, by improvements in the smelting practice, especially the adoption of the Rhenish furnace.

The continued high price for spelter has stimulated important experiments upon the treatment of low-grade calamine ore, with a view to the extraction therefrom of a rich zinc oxide, which can then be passed on to the smelting process. An experimental plant of this character is in operation at the Elizabeth dressing works. The Zinkgewinnungsgesellschaft m. b. H., of Berlin, has purchased a large dump of calamine tailings, containing about 10 per cent. zinc, which it is aiming to re-work metallurgically. The scheme which is being tried at the Elizabeth dressing works is based on the distillation of the ore. [Probably in grate-kilns as in the Wetherill process of oxide manufacture.—Ed.] The Berlin company, on the other hand, is aiming to effect an extraction of the zinc oxide by means of dilute acid and precipitate hydroxide of zinc by means of lime. Neither of these processes, up to the present time, has passed beyond the experimental stage.

The business in sheet zinc in Silesia was also very satisfactory in 1905. The production was about 54,000 metric tons. The companies which are engaged in the rolling of sheet zinc in Silesia are combined in a union, this including the mills at Lipine, Ohlau, Jedlitze, Piela, Hohenlohehütte, Schoppinitz, Antonienhütte, Kunigundehütte (at Myslowitz), and the Humboldt works at Kalk, near Cologne.

The Wetherill and Mechernich systems of magnetic separation are coming increasingly into use. An installation of this character is in operation at the smeltery at Lipine. For the concentration of low-grade zinc ore, the chemical process of Dr. Danziger is worthy of mention. In this process, pyritous zinc ore is treated by a roasting to convert the iron into sulphate, which can be leached out with water, leaving the blende behind.

For the roasting of zinc blende, there are at present in Silesia 12 plants, which have 128 furnaces, the gases from which are wasted; 144 furnaces, the gases from which are utilized for the manufacture of sulphuric acid; and 10 furnaces, the gases from which are utilized for the manufacture of sulphurous acid.

The mechanical roasting furnaces, which are so extensively in use in the United States, have not yet found wide employment in Silesia. The Bergwerksgesellschaft Georg von Giesche's Erben has six large mechanica furnaces in use. It was necessary, however, to make important modi-

fications in these furnaces before they could be successfully employed. The application of mechanical furnaces to the roasting of zinc blende without utilization of the gases for sulphuric acid manufacture is a good deal simpler than when the gases have to be employed for that purpose. After numerous experiments, a design of furnace was perfected wherein producer gas for the heating of the roasting-muffle could be employed with favorable economic results.

Some experiments have been made at the Rosamundehütte upon the employment of carborundum in the mixture for muffle manufacture. This gave favorable results. In general, in Silesia at the present time the muffles are made by hydraulic presses, as in good practice in zinc smelting elsewhere. The lining of these hydraulically pressed retorts with a special refractory material would appear to offer further advantages.

The Spelter Markets in 1905.

New York.—The violent fluctuations to which the spelter market is always subject again played an important part in the history of this year's business. While, in a general way, the level of prices was satisfactory, still the smelting industry did not profit largely, because the price of the raw material was most of the time relatively higher than that of the refined metal. This applies particularly to the Joplin orefield, where it would seem that the maximum production has been reached, while the demand for this class of ore has increased right along, owing to the growing consumption of high-grade spelter for brass and rolling mill purposes.

The consumption of spelter throughout 1905 was very satisfactory, and stocks which existed at the beginning of the year disappeared during subsequent months. Some exports were made in the fall, amounting to about 4000 tons, but it was not the question so much of a surplus of metal as the fact that the European markets had reached the parity of the American market, and the scarcity of spelter abroad necessitated imports from here.

At the close of the year the outlook was good for a continued heavy demand, galvanizers and brass manufacturers being very busy.

Prices at the beginning of 1905 were 6.05c. St. Louis and 6.20c. New York, and remained on this level during January. Ore prices in Joplin became somewhat softer at the beginning of February, in consequence of which spelter quotations receded 0.05c. per ℔. and remained steady thereat during that month.

The first half of March brought a large inquiry from consumers, so that prices hardened again. As soon as this demand was satisfied, however, the market again gave way, and as no support was forthcoming, declined gradually to 5c. St. Louis and 5.15c. New York, which level was reached the beginning of June.

These low prices attracted a good many buyers, who received further encouragement by an improvement in the galvanizing industry. Quotations at the end of July had reached 5.35c. St. Louis, and 5.50c. New York. A further impetus was given the market at the beginning of August by reports from Joplin of a falling off in the ore supply, which necessitated the dead-firing of a number of furnaces. The month closed at 5.60c. St. Louis, and 5.75c. New York.

The firmness of the metal was maintained during September and prices at the end of that month had advanced to 5.80c. St. Louis, and 5.95c. New York. In October considerable sales for export to Europe (where spelter had become so scarce that prices there were upon a parity with ours here) were made, which cleared out almost all of the stocks that were held by the smelters. Quotations were established at 6.05c. St. Louis, and 6.20c. New York.

Our market here, in consequence of the close proximity to London, became naturally dependent on the same, and as prices over there receded somewhat through the exports from this side, November brought us lower quotations and closed at 5.95c. St. Louis, and 6.10c. New York.

At the beginning of December the demand from both the brass and galvanizing trades became much more active, and the comparatively small stocks which had accumulated in the hands of the smelters were gradually absorbed. In consequence of this, the year closed with a very firm market, prices being quoted at 6.50c. St. Louis, and 6.65c. New York.

AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900...	4.65	4.64	4.60	4.71	4.53	4.29	4.28	4.17	4.11	4.15	4.29	4.25	4.39
1901...	4.13	4.01	3.91	3.98	4.04	3.99	3.95	3.99	4.08	4.23	4.29	4.31	4.07
1902...	4.27	4.15	4.28	4.37	4.47	4.96	5.27	5.44	5.49	5.38	5.18	4.78	4.84
1903...	4.87	5.04	5.35	5.55	5.63	5.70	5.66	5.73	5.69	5.51	5.39	4.73	5.40
1904 ..	4.863	4.916	5.057	5.219	5.031	4.760	4.873	4.866	5.046	5.181	5.513	5.872	5.100
1905 ..	6.190	6.139	6.067	5.817	5.434	5.190	5.396	5.706	5.887	6.087	6.145	6.522	5.882

AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN ST. LOUIS.

Year.	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1903..	4.688	4.681	5.174	5.375	5.469	5.537	5.507	5.550	5.514	5.350	4.886	4.556	5.191
1904..	4.673	4.717	4.841	5.038	4.853	4.596	4.723	4.716	4.896	5.033	5.363	5.720	4.931
1905..	6.032	5.989	5.917	5.667	5.284	5.040	5.247	5.556	5.737	5.934	5.984	6.374	5.730

Zinc Sheets.—The price of sheet zinc at the end of 1905 was $7.75 per 100 ℔. (less discount of 8 per cent.), f. o. b. cars Lasalle and Peru, in 600 ℔. casks, for gages No. 9 and 22, both inclusive, widths from 32 to 60 in., both inclusive, and lengths from 84 to 96 in., both inclusive. The freight rate to New York is 27.5c. per 100 ℔. The fluctuations in the base price for sheet zinc during 1905 were as follows: December 30, 1904, $7.25;

January 7, 1905, $7.50; May 12, $7.25; June 1, $7; July 29, $7.25; August 10, $7.50; December 1, $7.75.

London.—Like its immediate predecessor, 1905 saw a steady and persistent advance in values, due to the irresistible force of expanding industry in consumption. During the first week large transactions were reported from the Continent, at full prices, and for delivery extending up to midsummer. The English market, though less active, was also favorably influenced—particularly by the growing demand for galvanized iron. Consumers, while reluctant to buy far forward at the apparently high prices, were prudent enough not to leave incoming orders uncovered. On the other hand speculators were tempted to push sales—particularly for extended delivery in the hope that prices might relax by natural gravitation to a more normal level. Their efforts were temporarily successful in so far that January closed with ordinary brands £24 12s. 8d. to £24 15s. 4d. ex-ship, and specials at 2s. 6d. more, representing a fall of 7s. 6d. over the month.

February opened with less activity in galvanized iron and in brass and yellow metal, but labor troubles were threatening in several of the Continental coal-mining centers, with a possible curtailment of the output of spelter and the market consequently remained steady awhile. The collapse of the coal strikes in Silesia and Belgium gave further encouragement to sellers who, later in the month, accepted down to £24 5s. ex-ship. Thereafter the market was subjected to further attacks until prices were forced down to £24 for ordinaries and £24 2s. 6d. to £24 5s. for specials.

March opened with transactions down to £23 15s.; but the evident depletion of stocks counteracted the apathy of consumers, and a temporary advance of 2s. 6d.was recorded, only to be followed by renewed onslaughts by sellers, resulting in a drop to £23 7s. 6d. Thereafter a recovery set in, and the month closed with ordinary brands £23 15s., and specials £24 to £24 5s.

April was uneventful until about the middle of the month, when improved demand for sheet zinc, and an enlarged trade in galvanized iron, raised price of ordinary brands to £24 for early delivery and about £23 15s. for forward delivery; which prices ruled up to the end of the month, with specials 5s. extra.

May brought forth fresh attacks which forced the price down to £23 10s. An increasing demand from galvanizers did little more than improve prices by 2s. 6d. toward the end of the month, notwithstanding that Continental producers were holding for much higher prices; and the closing price was £23 12s. 6d. for ordinary brands, and £24 to £24 5s. for specials.

June revealed improved consumption on all sides, and a steady advance carried prices up to £23 15s. and £24, according to delivery. Specials were quoted £24 2s. 6d. to £24 5s.

July opened with accumulating evidence of large requirements. By the middle of the month ordinary brands commanded £24 2s. 6d. to £24 7s. 6d. Toward the close persistent pressure forced prices down to £23 15s.; but the month ended with a recovery to £23 18s. 9d., with specials at the relatively high figure of £24 10s. to £24 15s.

August found the inevitable advance manifest early in the month, and at the end foreign was quoted at £25 17s. 6d. and specials £26 7s. 6d.

September brought forth eager demand from all sides, notably from galvanizers, while prices of sheet zinc were raised repeatedly. There was little advance, however, until about the third week; thereafter prices advanced uninterruptedly to £27 7s. 6d. for foreign and £27 12s. 6d. for special brands.

October saw increased activity. Transactions were numerous and large, at successive advances, and extending over long periods. Foreign, by the middle of the month, commanded £28, which led to some transient business with America. But the prompt raising of American prices precluded further imports. The month closed with foreign quoted about £28 10s., and specials £28 15s. to £29.

November opened with consumers fairly covered for their early requirements. This caused a temporary relapse to £27 15s.; but this was quickly followed by further advances, and the month closed with foreign at £28 7s. 6d. and specials at £28 10s. to £28 15s.

December found the trade quiet, buyers being reluctant to pay ruling prices, and producers having practically nothing to offer. Business was meager in volume until toward the middle of the month, when considerable quantities changed hands at £28 15s. for ordinary brands. The market at the close was steady at £28 15s.

AVERAGE MONTHLY PRICE OF SPELTER IN LONDON.
(Pounds sterling per ton of 2240 lb. of good ordinary brands.)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£	£	£	£	£	£	£	£	£	£	£	£	£
1905....	25.063	24.594	23.825	23.813	23.594	23.875	23.938	24.675	26.375	28.225	28.500	28.719	25.433

Zinc Oxide.

Zinc oxide was produced in 1905 by the New Jersey Zinc Company and its subsidiary company, the Mineral Point Zinc Company; also by the Ozark Zinc Oxide Company of Joplin, Mo., and Coffeyville, Kan. The New Jersey Zinc Company supplies the eastern market with the oxide made at its works at Palmerton, Penn.; the Mineral Point Zinc Company supplies the market in the central and western States from its works at Mineral Point, Wisconsin. The Ozark Zinc Oxide Company supplies a portion of the market in Kansas City and the Southwest.

The Palmerton works employ the franklinite ore from Mine Hill, N. J. The Mineral Point Zinc Company uses ore partly from Wisconsin, and ore partly from Colorado and other districts west of the Rocky Mountains. The Ozark Zinc Oxide Company, which is a subsidiary of the Sherwin-Williams Paint Company, obtains its ore supply from Kelly, N. M., where it owns the Graphic mine. This company has recently completed a large plant at Coffeyville, in the Kansas natural gas field, and is planning to increase greatly its scope of operation.

The total production of zinc oxide in the United States in 1905 was 65,403 short tons, valued at $80 per ton f. o. b. works, against 57,613 short tons, valued at $78.50 per ton in 1904. The large annual increase in the production of this substance is chiefly indicative of the growing demand for it as a pigment, although a considerable part of it is consumed in the rubber trade, which is also increasing rapidly. The great demand for zinc oxide in the paint trade was one of the noteworthy features of 1905. This pigment is now used so extensively in mixed paints that the increase in its sales keeps pace with the increased sale of paint. Although the nominal value f. o. b. works shows only a small increase in 1905, the market quotations advanced 0.25c. per lb. during the year.

PRODUCTION OF ZINC OXIDE IN THE UNITED STATES.

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Ton.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1898	32,747	29,708	$2,226,796	$68.00	1902	52,730	46,929	$4,023,299	$76.30
1899	39,663	35,982	3,331,692	84.00	1903	59,562	54,034	5,005,394	83.69
1900	47,151	42,775	3,772,080	80.00	1904	59,613	54,081	4,523,414	75.88
1901	46,500	42,266	3,720,000	80.00	1905	65,403	59,349	5,232,240	80.00

The production of zinc-lead pigment by the United States Smelting Company, at Canon City, Colo., is not included in the above table. In 1905, this amounted to 7200 tons, valued at $540,000 ($75 per ton), against 6781 tons, valued at $474,670 ($70 per ton) in 1904.

Progress in the Metallurgy of Zinc.

By W. R. Ingalls.

Metallurgically, there was a good deal of experimentation in 1905, and some steady improvements in the practice, but nothing of a remarkable nature. Experiments were made with the Lungwitz process of smelting zinc ore in the blast furnace at Warren, N. H., on a rather elaborate scale. At Vancouver, B. C., experiments were made with an electric furnace invented by F. T. Snyder, but this process also is only in the experimental stage. At Denver, Colo., a considerable quantity of ore was actually treated by the Dewey process, which consists substantially

in obtaining a solution of the zinc of the ore in the form of sulphate, evaporating the sulphate to dryness and calcining it for the production of oxide. During the autumn the plant was partially destroyed by fire, but it is the intention to rebuild it in 1906.

In mechanical concentration further progress was made, and now it is indeed a difficult ore which cannot be enriched to a grade of at least 40 per cent. zinc in one way or another. In many cases ore of upward of 50 per cent. zinc can be produced. This is of course largely dependent upon the character of the blende in the crude ore. If it contains so much combined iron that the pure mineral assays only 60 per cent. zinc, for example, a commercial concentrate assaying 55 per cent. is about the highest that can be expected under any circumstances.

Composition of Ores.

According to W. George Waring the average of 30 recent shipments of blende from Jasper county, Mo. (Joplin district), showed 58.477 per cent. zinc, 1.853 per cent. iron, 1.37 per cent. lead, 0.36 per cent. cadmium, 0.03 per cent. copper, 0.0159 per cent. arsenic, 0.00073 per cent. phosphorus, and traces of manganese, nickel and cobalt. Antimony was not tested for. When each of the 30 samples was assayed for lead, using 3 grams for each determination, the entire filtrates and wash-waters from the precipitation of the lead sulphate were combined, and the cadmium, copper, arsenic, phosphorus and manganese were estimated in the entire liquid, representing 90 grams of ore.

In the separate samples the percentage of zinc ranged from 55.65 to 62.60; that of iron from 0.62 to 5.45, and that of lead from *nil* to 3.37. One sample of ore, assaying 61.95 per cent. zinc, showed 2 per cent. iron and no lead. A sample, assaying 62.60 per cent. zinc, showed 1.12 per cent. iron and 0.32 lead. A sample, assaying 61.75 per cent. zinc, showed 0.62 per cent. iron and 0.09 per cent. lead.

W. W. Petraeus stated in the *Lead and Zinc News* (January 15, 1906) that 600 shipments of blende from the Joplin district in 1897 averaged 56.64 per cent. zinc; 1299 in 1898 averaged 56.99 per cent. zinc; 3715 in 1899 averaged 56.67 per cent. zinc; 3500 in 1900 averaged 57.36 per cent. zinc and 0.22 per cent. iron; 3683 in 1901 averaged 57.53 per cent. zinc and 2.02 per cent. iron; 5776 in 1902 averaged 57.08 per cent. zinc and 2.24 per cent. iron; 6014 in 1903 averaged 57.30 per cent. zinc and 2 per cent. iron; 6220 in 1904 averaged 57.24 per cent. zinc and 2.1 per cent. iron; 4628 in 1905 averaged 57.97 per cent. zinc and 1.84 per cent. iron. The assay basis of settlement was not inaugurated until 1899. The shipments in 1905 assayed 1 per cent. lead, 0.001 per cent. copper, and 0.198 per cent. cadmium.

Mechanical Concentration.

The following methods are now available: (1) Hand-sorting. (2) Wet-jigging and tabling. (3) Dry-jigging and tabling. (4) Electro-magnetic separation, (*a*) on roasted ore, (*b*) on raw ore. (5) Electrostatic separation. (6) Flotation. The practice in, and general applicability of, these processes are summarized as follows:

1. Hand-sorting is applicable wherever the mineral occurs in lumps large enough, say in excess of 1-in. size, to be picked out as clean mineral. Wherever this condition exists, hand-sorting should be applied, inasmuch as it enables the production of high-grade concentrate without subjecting the mineral to the expense of further crushing and the danger of loss by sliming. Hand-sorting is not an expensive process, and it should be resorted to a good deal more extensively than is commonly the custom. It is practiced with great advantage in many of the best-managed zinc concentrating mills of the United States.

2. Wet-jigging and tabling is the standard method of ore concentration, and on account of its cheapness should be practiced wherever possible. Fine grinding and the introduction of the Wilfley table led to a great advance in this art. The mixed ores of Broken Hill and Leadville were originally treated in this way. When it is merely a case of mineral separation, as for example the production of a lead concentrate and of a zinc concentrate, each product of the mill may be discharged into a filter bin, where all the mineral is caught, the water filtering out through the burlap lining, and there is no loss.

3. Dry-digging and tabling has come into prominence through the success of the Sutton-Steele system of concentration, which was described by R. C. Canby in the *Engineering and Mining Journal* of May 12, 1906. The Sutton-Steele table has a porous deck, through which air is caused to pass under about 0.5-oz. pressure, which causes the ore to pass over the table just as hot roasted ore spreads over damp ground when dumped upon it. The table has a reciprocating motion, like the wet concentrating table, but apparently is able to accomplish greatly improved results, at least in some cases. This process is being installed at the Tiro General mine, in Mexico.

4. Electromagnetic separation is applicable for the removal of zinc from an ore, when the zinc mineral contains sufficient iron to be attractable by high intensity magnets, like the Wetherill, Mechernich, or International magnets. This presupposes the non-existence of other equally magnetic minerals in the ore. In other cases, as for example a mixture of marcasite, pyrrhotite or pyrite and blende free from iron, separation can be effected by roasting the ore in order to convert the pyrites into magnetic form, and then removing it by magnets of low intensity. A great range

of variation in magnetic separation is possible through modifications in the roasting, in the intensity of the magnets, etc.

5. Electrostatic separation is applicable in the case of two minerals, of which one is a conductor of electricity, and the other is not. Blende is commonly a non-conductor, while pyrites, galena, etc., are commonly conductors. There is, however, great variation in this property of minerals, and the precise conditions can be determined only by actual test. The chief separator of this description heretofore in use has been the Blake. In some cases it has been very successful; in other cases it has failed. The Hough separator is said to be giving excellent results in a large experimental way, but has not yet been placed on the market. A new machine, said to be giving promising results, is the Sutton-Steele dielectric separator, which, unlike other electrostatic separators, does not depend upon the difference in electric conductivity, but acts by inducing dielectric hysteretic impedance in the particles to be separated. This is induced by several different methods of interrupting, or reversing, the current, in conjunction with a convective discharge through the mass of material upon the separating roller. The apparatus and process are described by R. C. Canby in the *Engineering and Mining Journal* of May 12, 1906.

6. Flotation, invented in Australia, has clearly beaten magnetic separation for the treatment of the mixed sulphides of Broken Hill. In this process the ore, finely ground, is introduced into a bath of water, acidified with about 2 per cent. of sulphuric acid. The acid acts upon carbonates in the ore, especially carbonates of lime and iron, and evolves carbon dioxide gas, the bubbles of which attach themselves to the particles of blende and galena, and thereby give them a buoyancy, which causes them to float to the surface of the bath, where they collect as a black scum. The operation is conducted in a vessel like the ordinary spitzkasten, but its operation is just the reverse of the ordinary, because instead of the heavier material being drawn off from the bottom, in this case the lighter material, i. e., the gangue, is drawn off from the bottom, while the heavy sulphides, raised to the surface, float off with the overflow. They pass into a settling tank, in falling into which the bubbles of gas are disengaged, causing the sulphide particles immediately to settle; the liquor is then drawn off and is returned to the process.

Two processes are now in regular use, namely, the Potter, which employs sulphuric acid alone, and the Delprat, which uses sulphuric acid in connection with a solution of sodium sulphate, the latter being added in order to densify the bath. By increasing the specific gravity of the separating bath in this way, a better separation appears to result. There has been litigation between the owners of the Potter and Delprat processes, Potter claiming that Delprat is an infringer, but the case has not yet

been settled; it attracted much attention during 1905. In connection with this litigation, Professor A. K. Huntington, and Messrs. James Swinburne and Dr. G. Rudorf, have done much experimental work, the result of some of which was given to the public through the proceedings of the Faraday Society, republished in the *Engineering and Mining Journal* of Feb. 10 and 17, 1906. These papers constitute the first practical literature on this important subject, and have done very much to throw light upon the nature of the physical factors involved in it. It appears from their studies that the presence of carbonates in the ore is an essential, but with that proviso the process is likely to be of general application.

Modifications of the original flotation process are the Bavay, in which the ore is gasified directly with carbon dioxide, such as may be obtained from furnace gases; and the Elmore and Cattermole, in which oil is used in connection with gas, the process thus becoming a combination of the original flotation process and the Elmore oil process. Experiments on a large scale are being made with all of these processes at Broken Hill, and also in Great Britain on many kinds of ore. I may say from my own experiments on flotation in a simple acid bath that the results attainable by this process are very remarkable, and it is undoubtedly a very important contribution to the art of ore separation. There is, however, evidently going to be much litigation over patent rights. The separation of blende from gangue appears to be very much more successful than the separation of blende from galena. Complaints have been raised over some of the concentrate produced in Australia, because of its high content in lead. The process is only in its infancy; not until 1904 did we begin to have much knowledge as to its whys and wherefores. Before the end of another year, our information will certainly be more extensive.

Roasting Furnaces.

The new works of the Mineral Point Zinc Company at Depue, Ill., and of Hegeler Bros. at Danville, Ill., both of which are to produce sulphuric acid, will be equipped with the Hegeler furnace. At Depue the Grillo-Schroeder contact process will be employed; at Danville, the chamber process. At Caney, Kansas, Zellweger roasting furnaces have been installed; at Coffeyville, Kan., McDougall furnaces.

New furnaces of the McDougall type were patented in 1905 by W. R. Ingalls (U. S. No. 786,567); Frank Klepetko (U. S. No. 799,063); Frank E. Marcy (U. S. No. 802,007); August R. Meyer, (U. S. No. 804,751); Ottokar Hofmann (U. S. No. 809,953); F. J. Falding (U. S. No. 788,098); and Henry Howard (U. S. No. 804,227). All of these patents cover modifications, some of which, however, are very important. It is becoming more and more recognized that furnaces of this type are likely to be

very important for blende roasting, both in connection with the manufacture of sulphuric acid, and for roasting by direct heat.

A furnace with revolving hearth, of the type of the old Brunton calciner, was patented by Frank H. Trego (U. S. No. 798,844). This furnace is used in Wisconsin, in connection with roasting previous to magnetic separation.

Smelting Furnaces.

A good deal of attention is now being devoted to the electrothermic smelting of zinc ore. One process, that of de Laval, is already in regular operation at three works in Norway and Sweden; experiments are being made with other types of furnace both in Europe and America. The applicability of this method of smelting is largely a matter of the relative cost of the heat units finally utilized, as compared with the cost by the direct combustion of coal, but there are many practical details that must also be successfully developed. Among other things the condensation of liquid spelter, and not merely blue powder, is a problem that has not yet been successfully overcome.

The attempts to improve the method of zinc smelting are not, however, confined to the new field of electric heating. The old idea of smelting continuously in a blast furnace is still being followed up. Some interesting experiments have recently been made, but not yet with conclusive results. There are so many confusing factors that may enter into this problem that no opinion should be expressed without an exhaustive study of the conditions in the light of the latest knowledge presented by physical chemistry. Practically we are not yet any nearer success in this direction than we were many years ago.

In the meanwhile, so long as distillation must be conducted in the historic manner, the mechanical and commercial success of the operation depends upon the design and construction of the furnace, to afford the longest possible campaign and the minimum loss of heat; also upon the preparation of the retorts, with a view to making them as dense and durable as possible; and on details in furnace management, of which some system of mechanical charging, an improvement that has long been hoped for, appears the most promising.

Furnaces.—Many types of furnace construction are in use, including some that date back three decades or more, but the Rhenish type in its several modifications appears now to have become the most approved form. This furnace is built with regenerative checker-works for both gas and air, as at Engis, Prayon, Overpelt and Pueblo; with regenerative checker-works for air only, as at Monteponi, although the Ferraris furnace employed there does not belong strictly to the Rhenish type; and with counter-current heat recuperation, as at Stolberg and Palmerton. All

of these furnaces afford long campaigns and are economical in their consumption of coal. It is difficult to say definitely which one is the best, since they do not often receive competitive trials under the same conditions.

The regenerative furnaces of the Siemens system are undoubtedly the more economical of coal, but the advantage in that respect may be offset by the increased extraction of metal that is effected by the steadier temperature of a well designed counter-current system of heat recuperation. The difficulty of keeping the flues of the latter type tight, under the severe expansive strains of the furnace, is an inevitable drawback, and on the whole they must be pronounced inferior to the furnaces which have Siemens checkers and employ reversing-regeneration either through the medium of gas and air, or of air alone.

Nevertheless, the Convers & de Saulles furnace has been decided the favorite at Palmerton, Penn., and in the extension of that plant all of the new furnaces are to be of this type. The small number of retorts (200 per block) is commendable; there is no advantage in greatly increasing the number of retorts per furnace, and there are many positive disadvantages. On the other hand, the new plant of the Mineral Point Zinc Company, at Depue, Ill., is to have six Neureuther-Siemens furnaces, with 800 retorts each. These furnaces are employed at Peru, Ill., but they do not represent the most efficient development of the Siemens system.

In the Hegeler furnace, used at Lasalle, Ill., the gases (including air) are introduced at the end and along the combustion chamber. A pressure above atmospheric and nearly uniform throughout the whole length of the chamber is desired and movement of the gases through the chamber is required. Resistance to the movement of the gases increases with the length of the chamber, so as to necessitate a pressure so great in the chamber near the entrance end as to be detrimental there. Experience in the use of the furnace proved that the upper tier of retorts are heated somewhat too much. Relief from this was obtained by contracting the space in the flue above the upper tier of retorts by lowering the rear of the top arch to a little below the upper line of the top tier of retorts at their rear end (making recesses in the arch for them). Such method of reducing the heat of the upper retorts, however, carries with it an increase of the resistance to the passage of the fire-gases through the chamber.

Other changes in the construction of the furnace from the original design (1879) had previously been made, viz.: First, the perforated retorts to serve for the introduction of air into the chamber and placed between the retorts charged with ore were omitted, thereby creating open vertical spaces between the several sections of four ore-retorts each; and, second,

the floor below the bottom tier of retorts was lowered to give space for droppings from the retorts; but dams of sand were made on the floor at every section of four retorts of a hight sufficient to project a little into the vertical spaces between the sections of retorts to obstruct the draft. It was also found that in a furnace consisting of 18 sections of four retorts each horizontally and six rows high the sand dams could be omitted under the first seven sections without showing bad effects on the uniformity of the heating.

E. C. Hegeler, in U. S. patent No. 794,799 (July 18, 1905), now describes an improvement, whereby means are provided for reducing the resistance to the gases in their passage through the combustion chamber without interfering with the uniform heating of all the retorts, a greater uniformity of the gases in the upper and lower parts of the retort chamber being secured. In this improvement, the main arch of the furnace is built in sections, separated by gaps equal to the width of four retorts in position. These gaps are closed in by arches, of which the axes are at right angles to the axis of the main arch. On the floor of the combustion chamber, immediately under these subsidiary arches, a baffle wall, or a dam of sand, is arranged. These modifications compel the gases to follow a slightly wavy line through the combustion chamber. It is said that this modification not only gives greater uniformity of the gases in the combustion chamber near the entrance, but also increases the heat near the working front of the furnace, and reduces it at the rear wall, on which the rear ends of the retorts are supported. The specification of patent also describes improved means for mixing the air and gas.

Herman and Julius W. Hegeler have patented (U. S., 792,773, June 20, 1905) a modification of the Siemens-Belgian furnace, which resembles the Neureuther furnace in having vertical channels in the middle wall; but these channels communicate only with the gas checkers, the air checkers communicating with the combustion chamber in the usual manner of this type of furnace. The effect of the Hegeler modification is to introduce the entire supply of air beneath the lowest row of retorts, the gas being introduced through ports in the middle wall at intervals higher up. This arrangement effects, therefore, a combination of the principle of the long furnaces used at Lasalle, with that of the Siemens furnaces used at Peru. Its purpose is of course to maintain an equal temperature throughout the combustion chamber, heating the lowest and highest rows of retorts to substantially the same degree. Openings are arranged in the arch of the furnace, in the conventional manner, for clearing out obstructions in the gas channel. The new furnace has not yet been tried practically.

Emile Dor-Delattre, in U. S. patent No. 806,121 (Dec. 5, 1905) describes an improved form of Siemens-Belgian furnace. The middle wall has a

series of large ports, below the lowest row of retorts, through which the products of combustion pass from the combustion chamber on one side to the combustion chamber on the other side, instead of passing over the top of the middle wall, below the arch, as in the usual furnace of this type. On top of the furnace are two air flues and two gas flues, extending longitudinally, communication being made with them through alternating slots in the arches of the combustion chambers. The regenerative chambers, both for air and gas, are situated at each end of the furnace. It is claimed that by this construction the air and gas ports are made more accessible for cleaning, wherefore they can be adjusted more closely to the quantity of gas that they must pass, thereby effecting economy in the consumption of gas.

Zinc Smelting in Silesia.—Attention has been repeatedly called to the change which the zinc smelting practice has been undergoing during the last 15 years. Judging from the statistics for 1905, the time is not far distant when the historic Silesian method will become obsolete, the Rhenish furnace having already attained a remarkable extension in its application. The new works erected in Upper Silesia are invariably equipped with that type of furnace, while the established works remodel their old furnaces at every opportunity. The prime cause for this departure in practice is the change in character and the increase in the average grade of the ore furnished by the mines, which is due to the exhaustion of the comparatively low-grade calamines and the increasing output of the sulphide ore, much richer in zinc.

Roasted blende is smelted more advantageously in small muffles than in large ones; in realizing the necessity for a change in their practice the Silesian smelters naturally turned to the Rhenish furnace, which has proved so successful in Rheinland and Westphalia, and from there has forced its way into Belgium and America as well as into Silesia. The Rhenish furnace itself is a modification developed by a combination of the best features of the original Belgian and Silesian types of furnace, dating from about 50 years ago, when the Silesian practice had a great influence upon that of the west of Europe. The changes which then emanated from Silesia are now reflected back to their source.

Zinc smelting was begun in Silesia in 1798; it was begun in Carinthia about the same time; but the art was not discovered in Belgium until 1807. It had been carried on in England, however, since 1740, and it is believed that the process employed there was introduced from China. So far as the historical evidence goes, these discoveries were independent. At all events, a radically different type of furnace was devised in each case. Of these, the Carinthian was the most inefficient, and after about 30 years it passed out of use, which never had been extensive. The English furnace survived, with diminishing use, until about 1860.

The Silesian furnace was evolved from a glass-melting process with which Johann Ruhberg made his first experiment, and the type has ever since retained certain of the original characteristics. After its introduction in the west of Europe, the single tier of large muffles was replaced by the two or three tiers of smaller muffles, which developed into the Rhenish furnace. This did not, however, establish itself in Silesia until recently. The Guidottohütte, built in 1891, was equipped with furnaces having two rows of small muffles, but they did not prove successful, and were rebuilt. In 1893 a similar furnace was tried at the Hohenlohehütte with more promising results. In 1898 the Rhenish furnace was introduced at Antonienhütte, where the charge smelted consisted of 70 per cent. roasted blende and 30 per cent. calamine, and since then the change to the new type has been rapid.

This change is of great commercial importance. The mines of Upper Silesia contained what were probably the largest deposits of calamine that have been known. The deposits of blende, below the line of oxidation, are believed to be even more extensive. The blende assays higher in zinc than the calamine, wherefore in producing the same tonnage of ore the quantity of zinc afforded is considerably increased. The quantity of zinc extractable is further increased by the adoption of the new method of smelting, whereby the percentage of loss is reduced. As a matter of fact, the tonnage of ore produced by the mines is increasing year by year, so that taking all the factors into consideration the large and steady increase in the smelter output of Silesia is readily explained. The chief limiting factor appears to be the legal requirement that the gases from the roasting furnaces shall be taken care of, which means practically the manufacture of sulphuric acid, and that is hardly a profitable by-product under the existing conditions.

Smelting in the Blast Furnace.—The experiments with the Nagel process at the Grillo works, Hamborn-Neumühl, Westphalia, have been abandoned, without reaching conclusive results. The experiments made with the Lungwitz process at Warren, N. H., during the autumn of 1905, have been referred to by F. W. Gordon in the *Engineering and Mining Journal,* Apr. 28, 1906. The furnace was designed on the lines of an iron blast furnace, being 3 ft. in diameter at the hearth, 5 ft. 5 in. in the bosh, and about 30 ft. high, and designed to withstand a working pressure of 100 ℔. per sq. in.; in fact, it proved tight at 120 ℔. Some difficulties were encountered, owing in part to newness of the plant and its design, and to want of familiarity with the unpracticed art, wherefore late in the season it was decided to defer further operations until this spring.

Armstrong, in British patent No. 20,543, of 1904, shows an improvement of his proposed blast furnaces, previously described, but so far as I am aware none of his designs has been tried practically.

The furnaces of P. Schmieder (which is now being tried in Silesia) and F. Kellerman are rather vertical-muffle furnaces than blast furnaces. The Schmieder furnace is a combination of vertical retort and shaft furnace. In the upper part of the column, which is the longer part, the retort is heated by gases burning in flues which surround it, the zinc vapor being taken off into lateral, horizontal condensers. The nearly spent residue passes down into the lower portion of the shaft, where hot air is introduced through tuyeres for the purpose of burning off the remaining zinc, fusing the slag, matte, etc., which are tapped off through a hole at the bottom. The idea does not appear to be very practicable.

The furnace described by F. Kellermann consists of a vertical cylinder, about 20 in. in diameter, the bottom of the cylinder being inclined toward an opening, in order to facilitate the discharge of the residue. The cylinder is heated from an external fireplace, the gases burning in flues surrounding the cylinder. The outlet for the zinc vapor is located in the lower part of the furnace. This design of furnace does not appear to be very promising.

According to Kellermann, blast furnaces promise very little success, because the dilution of the zinc vapor with inert and oxidizing gases is too great to permit of their condensation. He figures that, in the zinc retort, 1 kg. of zinc in the charge produces about 172 liters of zinc vapor. In using 0.2 kg. of carbon for reduction of the oxide, there is produced 342 liters of carbon monoxide, wherefore the relative volumes of the zinc vapor and carbon monoxide are 1:2. For the smelting and reduction of the ore in the blast furnace, with an estimated utilization of 60 per cent. of the calories of the carbon, 1 kg. of zinc would require 0.6 kg. of carbon, which in burning to monoxide would require 2700 liters of air and would produce 3264 liters of combustion products (carbon monoxide and nitrogen). The ratio of zinc vapor to combustion products would be therefore 1:19. In practice the percentage of zinc vapor discharged by the blast furnace would really be smaller. This great dilution of the gas is one of the obstacles to success in attempts to smelt zinc ore in the blast furnace.

Retorts and Condensers.—In the standard practice of smelting, experiments are directed toward the production of a more durable retort. It has been found that the addition of a certain proportion of carborundum or siloxicon to the batch for the manufacture of the retorts is highly beneficial. However, the high cost of carborundum or siloxicon is temporarily prohibitive of its use in that way. There is promise, however, that the difficulty can be overcome by using these materials as a lining for the retort made by hydraulic pressure in the standard manner. A process of doing this has been invented by A. L. J. Queneau, and has been tried practically by the New Jersey Zinc Company, at Palmerton, N. J., with very encouraging results.

The experiments in Europe with retorts made with carborundum, or

siloxicon, in the batch, have given very successful results as to life, the latter extending up to 300 days. However, it is to be pointed out that very great durability is not the sole consideration, because after a certain time the interior walls will become so crusted up with slag and accretions that the retort will have to be discarded for that reason.

Miscellaneous.

A mechanical charging machine has been invented by A. L. J. Queneau (U. S. patent No. 813,022) and tried practically at Palmerton. It shoots the mixed charge into the retorts, the apparatus being designed for raising and lowering to correspond with the position of the retorts in the furnace, the whole being arranged on a truck running on rails in front of the furnace. The machine is large and heavy, but it injects the charge solidly into the retort in an efficient manner. It will doubtless effect savings in the furnace work, but to just what extent remains to be proved. So far the machine has only been tried experimentally, and, pending improvements which suggested themselves in these trials, it has not yet been introduced into the regular work.

The smelting of retort residues made from Western zinc ores, as practiced by the Cherokee-Lanyon Spelter Company, at Gas, Kan., in 1903, was described by E. M. Johnson, in the *Western Chemist and Metallurgist*, August, 1905. The furnace was 36x90 in. at the tuyeres, with a 10-in. side bosh and a 6-in. end bosh. The hight from the tuyeres to the charge floor was 14 ft. Although the residues comprised a large proportion of coal and coke, it was impossible to reduce the percentage of good lump coke on the charge lower than 12.5 or 13. The amount of charge smelted per day was about 45 tons. The charge contained 35 per cent. of zinc residue by weight, but upward of 50 per cent. by bulk. After the furnace was in good running order, the saving of silver was 90.8 per cent.; of gold 92.4 per cent.: of lead, 92 per cent.

The Bertha Mineral Company, at Austinville, Va., is treating low-grade calamine (15 to 20 per cent. zinc) on Wetherill grates, producing an oxide containing 70 to 75 per cent. zinc and 4 per cent. lead, which is shipped to Pulaski, where it is smelted for spelter in the usual way. A similar process is being tried in Upper Silesia. It is an entirely practicable method of treating low-grade ores. The only question is the rentability, which of course will vary with the conditions. Artificial oxide of zinc produced hydrometallurgically is one of the easiest products to reduce and distil. Oxide produced pyrometallurgically, on the other hand, is reduced with difficulty, the intensity of the conditions under which it is formed apparently binding the zinc and oxygen together with great force.

The Pape-Henneberg process is to be employed by the International Metal Company, Ltd., at Hamburg, Germany. The essential feature of

this process is a preliminary distillation of the ore in a special furnace, the zinc being collected therefrom in the form of oxide. The oxide is then to be smelted for spelter in the usual manner. It is contemplated to smelt low-grade and mixed ores from Australia, Great Britain, Upper Silesia and elsewhere. The process has already been tried experimentally under good supervision.

The process invented by A. H. Imbert (U. S. patent No. 807,271, Dec. 12, 1905) consists in smelting zinc and lead sulphides with granulated copper, when the reactions $PbS + 2\ Cu = Cu_2S + Pb$, $ZnS + 2\ Cu = Cu_2S + Zn$ take place, the former at 800 deg. C., the latter at 1100 deg. The copper is recovered for further use by bessemerizing. The process has been tried experimentally in France, with successful results as to the reactions, but its rentability has not yet been worked out.

August R. Meyer patented a process (U. S. 815,614, March 20, 1906) on somewhat the same line.

The number of proposals for hydrometallurgical processes, including chemical and electrolytic precipitation, was smaller in 1905 than usual, but nevertheless was considerable. Few of these, if any, merit special attention.

Anson G. Betts, of Troy, N. Y., proposes (U. S. patent 791,401, May 30, 1905) to extract zinc from its ores by first obtaining the metal as a soluble salt, such as zinc sulphate, in solution, the latter being then electrolyzed, using a liquid-metal cathode, such as mercury, in order to produce a zinc-mercury alloy. The zinc is extracted from the alloy, using a solution of zinc chloride as the electrolyte.

Electrothermic Smelting.

The de Laval process continues in regular operation at Trollhattan, Sweden, and Sarpsborg, Norway, about 2500 h.p. being used at the former place and 4000 h.p. at the latter; but as to its commercial success no information has yet been made public. The products first produced are blue powder and spelter high in lead. Redistillation gives the high grade of spelter which is marketed. Many patents on electric furnaces for zinc smelting (among others by W. McA. Johnson, U. S. 814,050, and F. T. Snyder, U. S. 814,810) have been taken out during the year, but none appears to promise any material advance in the art. Aside from the question of operating cost (power, labor, repairs, etc.) there are serious metallurgical difficulties, prime among which is condensation of the zinc as metal, not as blue powder, which no one, including de Laval, appears yet to have accomplished in ore smelting.

Composition of Spelter.

With respect to American spelter, most of the smelters of Kansas now

use ore from Colorado and elsewhere in the Rocky Mountains. Some of them use it exclusively. The ordinary brands of Western spelter that are now on the market consist, therefore, largely of metal derived from these ores. This is, however, as good for all practical purposes as the metal smelted from ordinary Joplin ore, inasmuch as at the temperature at which the ore is distilled no excessive quantity of lead is driven over; while if the ore be distilled at a high temperature, driving over a considerable percentage of lead, the excess can be removed by a simple process of refining. Spelter made from Colorado ore need not contain more than about one per cent. of lead.

It is the practice of nearly all the smelters to reserve the first draw of metal for marketing as special brands, these commanding usually a premium of 10c.@15c. per 100 ℔. This metal comes over during the first part of the distillation, before the temperature of the furnace attains the maximum, wherefore it is lower in lead than the average, but on the other hand is higher in cadmium. There has been a good deal of discussion among the smelters and brass makers as to the effect of a high cadmium content on spelter. Some of the specials produced in Kansas assay as high as 0.4 per cent. in cadmium. No definite conclusion has yet been reached as to the effect of this impurity. The consensus of opinion appears to be, however, that it is ordinarily not injurious, but if it exceeds 0.5 per cent., which is not often the case, it may detract from the value of the spelter for certain uses.

The effect of a small cadmium content in spelter was discussed in the *Engineering and Mining Journal* (April 13, 1905, p. 697). A further contribution on this subject has been made by F. Novak, in a paper entitled "Physical-Chemical Studies Respecting Cadmium in Lead-containing Zinc" (*Zeit. f. anorg. Chem.*, 1905, XLVII, 421). The most important results of his investigations are that an increase in the cadmium content of spelter rolled at 120 deg. C. effects a reduction of the reaction-velocity in the solution of the spelter in dilute chlorhydric acid; but an increase in the case of dilute nitric acid. The addition of 0.25 per cent. of cadmium to spelter gives the latter, after rolling and recrystallization by strongly heating, a finer grained structure than the cadmium-free zinc. The addition of 0.25 per cent. of cadmium to zinc increases its hardness and tenacity, and diminishes its brittleness. A higher content of cadmium (over 0.5 per cent.) makes the spelter softer, more brittle, and weaker than pure zinc; and consequently has an unfavorable effect on the quality of the metal.

Analytical Methods.

The standardization of methods for the determination of zinc was a subject of much discussion in 1905, and several committees appointed by chemical societies made reports. The matter is not yet settled, but up to

date the report of H. Nissenson (chief chemist of the Stolberg-Westphalia company) and W. Kettembeil is the most conclusive.

At the last International Congress of Applied Chemistry at Berlin, in 1903, it was resolved to appoint a commission to collect all methods of determining the most important metals; and, after examining their relative value, to recommend for international use those which were considered best. The task of carrying out this work for the metal zinc was assigned to Nissenson and Kettembeil.

In their report[1] they state that in the case of zinc it seemed particularly desirable to revise the methods hitherto in use, and, if possible, to secure uniformity. The gravimetric method is lengthy and difficult. Of the volumetric methods, two have found about equally general application; each offers certain advantages, but also certain disadvantages. After inquiring what methods were commonly used at different works, it was decided to test experimentally the value of the chief of these, comparing them with the gravimetric hydrogen sulphide method. The experiments show that the results obtained by the several methods are equally good. The electrolytic method also gives good results but cannot be used at most works, owing to the absence of the requisite equipment. The report concludes thus:

"Both with Schaffner's method and the ferrocyanide method, care will have to be taken that the same conditions are observed throughout. Contrary to the statements of Koninck and Prost, we have not found any influence of the oxidizing agent upon the end-point. As regards the indicator in this method, we find that uranium nitrate or ammonium molybdate can be used equally well. For weak eyes, the latter seems to be more agreeable and more certain. The end-point is reached about 0.2 c.c. earlier with molybdenum; it also has the advantage that it shows itself at once, and in its full power, whereas in the case of uranium there is a gradual darkening.

"As regards Schaffner's method, we have come to the conclusion that its best form is that used by us; while the Belgian method, using $2\frac{1}{4}$ gm., has the disadvantage (especially if much iron is present) that it is apt to give too low results. The titration with two burettes should be unnecessary for an experienced analyst, though it may do good service in the hands of the beginner while he is making himself familiar with the method."

[1] *Chemiker Zeitung*, 1905, LXXIII, pp. 951-955; *Engineering and Mining Journal*, Nov. 25, 1905.

PROGRESS IN ORE DRESSING AND COAL WASHING IN 1905.

By Robert H. Richards.

Crushing Machinery.

Sturtevant System.[1]—L. H. Sturtevant describes his method of crushing, including roll jaw-breaker, balanced rolls and centrifugal finishing rolls, but disclaims that they can compete where amalgamation is to be done with stamps, Huntington or Chili mills.

The Effect of Variations in Speed of Crushing upon the Production of Undersized Material.[2] —H. W. Gartell gives the results of his experiments in crushing at various speeds to see if increasing the speed of crushing increased the fines. The ores used were: a tough galena-blende ore with a silicious gangue and considerable feldspar, and a clean cobbed galena ore. The machines used were: a Krom jaw-breaker (5x12 in.), at speeds of 73, 285, 407, and 520 r.p.m.; a Gates breaker, style F, at speeds of 136, 538, 740, and 980 r.p.m.; and a pair of Krom rolls (16 in.) at speeds about 20, 85, 115, and 145 r.p.m. The results of screening the different products are given in tabular form, of which the following is an example:

SCREEN-CLASSIFICATION OF ORE CRUSHED IN JAW-BREAKER.

Mean Screen Aperture in Millimeters.	Per Cent.				Cumulative Per Cent.			
	520 R.P.M.	407 R.P.M.	285 R.P.M.	73 R.P.M.	520 R.P.M.	407 R.P.M.	285 R.P.M.	73 R.P.M.
44.	.8	2.1	1.8	2.4	.8	2.1	1.8	2.4
32.	17.5	15.5	20.1	24.9	18.3	17.6	21.9	27.3
21.	34.7	35.7	34.6	30.9	53.0	53.3	56.9	58.2
15.2	16.8	16.7	14.5	14.8	69.8	70.0	71.0	73.0
10.3	10.2	9.9	10.8	10.5	80.0	79.9	81.8	83.5
6.7	5.3	5.7	4.8	4.4	85.3	85.6	86.6	87.9
5.2	2.2	2.1	1.9	2.1	87.5	87.7	88.5	90.0
4.0	2.0	1.6	1.8	1.5	89.5	89.3	90.3	91.5
3.46	.9	.9	.8	.8	90.4	90.2	91.1	92.3
2.58	1.7	1.7	1.5	1.5	92.1	91.9	92.6	93.8
1.50	1.9	2.0	1.8	1.4	94.0	93.9	94.4	95.2
1.05	1.2	1.2	1.0	.9	95.2	95.1	95.4	96.1
.71	.8	.8	.8	.7	96.0	95.9	96.2	96.8
.51	.8	.7	.7	.6	96.8	96.6	96.9	97.4
.16	1.5	1.5	1.4	1.2	98.3	98.1	98.3	98.6
Less than .16	1.7	1.9	1.7	1.4	100.0	100.0	100.0	100.0
Weight of ore in lbs.	284	354	248	297	284	354	248	297

With the cumulative percentages as ordinates and with the logarithms of the sieve apertures as abscissæ the author plots curves illustrating his results in a graphical way. The author comes to the conclusion that, while decrease of speed tends to decrease the production of undersized

[1]*Mining Magazine*, Vol. XI (1905) p. 501.
[2]*School of Mines Quarterly*, Vol. XXVII (1904-5) p. 28.

material, it only does so in a degree so small that in general it is altogether disproportionate to the disadvantage of decreased capacity. Moreover, the range, both in speeds and in tenacity of material, was sufficiently great to render it probable that similar results would be obtained under almost all conditions.

Experiments in Crushing Gold Ores.[3]—W. Fischer Wilkinson gives the results of his experiments in crushing ore from the Simmer and Jack and Robinson Deep mines, with Edison rolls, 24 in. diameter, 6 in. faces. The following table is the result of the tests:

ORE-CRUSHING EXPERIMENTS.

No. of Test.	Air Pressure lb. per sq. in. Air Cylinder.	Quantity of Banket Ore Crushed.		Material Screened per H.P. Hour in lbs.	% Passing Screen Holes .018x.25 in.	% through 60 Mesh.	% through 90 Mesh.
		Per Hour in Tons, 2000lb.	Per H.P. Hour in lbs.				
1	30	44.4	2,056	212	10.3	52.7	34.0
2	30	52.5	2,540	254	10.0	49.1	37.4
3	30	50.0	2,610	217	8.3	49.8	35.8
4	20	52.3	3,020	278	9.2	49.2	35.2
5	20	53.1	3,008	289	9.6	47.7	35.1
6	20	57.0	3,240	269	8.3	44.2	32.8
7	10	49.0	2,890	228	7.9	46.8	36.9
8	10	54.4	3,100	229	7.4	41.9	36.8
9	10	59.9	3,522	211	6.0	45.1	34.9
10	30	54.1	2,504	260	10.4	48.9	36.9
11	30	56.7	2,624	273	10.4	49.7	36.8
12	30	43.4	2,594	259	10.0	47.2	34.9
					Holes .015x.25 in.		
13	30	63.6	3,270	353	10.8	54.1	36.3
14	30	65.8	3,340	307	9.2	54.1	36.3
15	30	64.8	3,330	270	8.1	54.3	36.1

The ores were graded before and after crushing. The following is a typical example:

Experiment No. 10.—Quantity crushed, 993 ℔.; air pressure, 30 ℔.; r.p.m., 200; time, 33 seconds; h.p., 43.2; calculated quantity crushed per hour, 54.1 short tons; quantity crushed per h.p. hour in pounds, 2504; quantity screened, 104 ℔.=10.4 per cent.; quantity screened per h.p. hour, 260 ℔.

Grade.	Crushing.	
	Before.	After.
On 2 mesh	18.8%	2.7%
Through 2 mesh on 5 mesh	51.3	42.4
" 5 " " 10 "	18.0	26.0
" 10 " " 12 "	2.7	4.2
" 12 " " 20 "	5.7	7.3
" 20 " " 30 "	3.1	7.1
" 30	0.4	10.2
	100.0	99.9

Dry Crushing of Ores by the Edison Process.[4] —W. Simpkin and J. B. Ballatine discuss the Edison crushing plants at Ogden, New Jersey, at Stewartsville, U. S. A., and at Dunderland, Norway. The Ogden plant

[3] *Trans.* of the Institution of Mining and Metallurgy, Vol. XIV (1904-5), p. 74.
[4] *Trans* of the Institution of Mining and Metallurgy, Vol. XIV (1904-5), p. 63.

has since been dismantled. The Stewartsville plant is used in crushing limestone and shale rock for making cement and subsequently in grinding the cement clinker. In a test run at this plant 50 tons of lime rock were ground in 53 minutes, 90 per cent. of the product passing a 200-mesh sieve. The ore at Dunderland consists of magnetite and specular hematite in a quartz gangue, and runs 40 per cent. iron. The machinery at the crushing-house consists of six sets of crushing rolls, the top set being 7 ft. diam. and 7 ft. face, and the others 5 ft. diam. and 3 ft. face. These rolls will crush pieces of rock weighing 8 tons to ½ in. cubes. This crushed material passes through a dryer house to the grinding-house, where it reduces from ½ in. to 25 mesh. The material from grinding rolls goes to the screen-house, where it passes over stationary screens. The slope of the screens is a little over the angle of repose, in this case 40 deg. From here it goes to the magnetic separation branch. A test was made with the grinding rolls on the Dunderland ore. 1000 lb. passed through the rolls in 20 seconds=90 tons per hour, using 34.9 h.p. The following results were obtained:

	Before	After
2 mesh and over	24.5 %	1.5 %
2 mesh to 5 mesh	34.00	32.5
5 " " 10 "	14.66	17.1
10 " " 20 "	12.5	12.5
20 " " 30 "	13.7	10.1
30 " and under	.6	26.3
	99.96%	100.00%

Rolls.[5] —R. K. Humphreys says rolls are used where the crushed grains of ore should be as nearly even in size as possible. Too great reduction by one machine should be avoided. The special points in design which he emphasizes are, sufficient strength to withstand the hardest ore, heavy cores or hearts to prevent jumping, springs in cages exerting no pressure on the journals when no ore is going through the rolls, forged steel tires so true inside and out as to wear down to 50 or 75 lb., hearts designed for rapid replacement of old tires by new, as few bolts to work loose as possible, and no sliding parts to wear out. He gives an instance of a tire that wore from 600 lb. to 50 lb.

The points in adjustment he emphasizes are means to keep faces parallel without unnecessary friction, journals running cool and having plenty of lateral motion to prevent ridges and flanges and make a uniform product, a releasing device to allow the feed to fall through till crushing can be resumed, and uniformity of hearts and shaft to make quick replacements. A roll that does not allow 80 to 90 per cent. of the tire to be worn away is poorly designed or operated.

In regard to rates of speed, in the first rolls crushing ore through ¾ in. to 1 in., to not finer than ¼ in., the peripheral speed should be 300 to 400 ft.

[5]*Engineering and Mining Journal*, Vol. LXXIX (1905), p. 77.

per minute, allowing the lumps to crumble and spread out, and not snap and jump. The second rolls, crushing oversize of 10-mesh, No. 20 wire screen, should make 800 to 1000 ft. peripheral speed per minute. The finishing rolls, receiving oversize 20 to 30 mesh, should make 1500 to 2500 ft. per minute.

In regard to capacity of rolls, with sufficient elevating and screening, three sets of 36x16 in. rolls at above speeds should crush 200 to 250 tons ordinary ore per 24 hours, or 350 tons oxidized quartz to 16 or 20 mesh, with a return from screens not to exceed 10 per cent.

Rolls Without Springs.[6] —Lewis Searing advocates no springs for crushing rolls. They add 15 to 20 per cent. to the cost of the machine, and when tension is neglected cause excess of oversize product, while rigid rolls prevent oversize, increase capacity and diminish cost, and prevent the pounding and jarring of the mill. The belts cannot bring power enough to injure rigid rolls if properly designed. A wooden key in the driving pulleys may be used as a precaution.

The Crosby Pebble or Ball-Mill[7] is a small testing laboratory grinder with two cylinders 10 in. diam., 12 in. long, with porcelain lining, lids and pebbles. It is very handy to charge, discharge and run.

Crushing Calcareous Tufa.[8] —In the answers to the question what machines are best for crushing this material, one writer advocates rolls, while another prefers Chili mills.

Huntington Mills.[9] —H. G. West writes concerning the Huntington mill as used at El Oro, Mexico, fifteen Huntington mills (diameter not stated) crush 22 metric tons from ¾ in. diam. to pass through a screen with 60 meshes to the linear inch in 24 hours. The wearing parts of Midvale rolled steel lose four pounds per metric ton crushed. The cost of repairs and renewals is 34½c. gold per metric ton. The housing has to be renewed every 12 months.

Chili Mills.[10]—It is recommended that the wheels of Chili mills be inclined inwards, so that the centrifugal force may all be spent in crushing.

Herr Amende-Braunschweig[11] illustrates the different forms of Chili mill used by clay workers.

Cone Mills.[12]—A description is given of the various designs of machines of the coffee-mill type for crushing burnt gypsum.

Williams' Hinged Hammer Coal Crusher[13] has a horizontal shaft making 600 to 1000 r.p.m., according to the size of the shaft, and has a series

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[6] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 657.
[7] *Clay Worker*, Vol. XLIV (1905), p. 141.
[8] *Tonindustrie Zeitung*, Vol. XXIX (1905), p. 132.
[9] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 1099.
[10] *Ibid.*, Vol. LXXX (1905), p. 932.
[11] *Tonindustrie Zeitung*, Vol. XXIX (1905), p 1041.
[12] *Ibid.*, Vol. XXIX (1905), p. 74.
[13] *Mines and Minerals*, Vol. XXV (1904-5), p. 391.

of heavy steel disks, each with four heavy steel hammers attached. The cylindrical housing is protected by heavy wearing plates, easily renewable. These plates on the under side have slots which serve as a screen to let the broken coal through. The machine is made in eight sizes, with a capacity of from 10 to 200 tons per hour breaking run-of-mine coal to $1\frac{1}{2}$in., 1 in., $\frac{1}{2}$ in., $\frac{1}{4}$ in., as preferred. They are used in New York, Pennsylvania and Virginia.

Jeffrey Crusher[14] is a hammer pulverizer which resembles the last.

Ball-Mills.[15]—A ball-mill with elevator, with wind separator for taking out the fines and returning the coarse to the mill, is described. Its duty is 10,000 to 12,000 ℔. to cement fineness, presumably in a working day.

The Wet Ball-Mill,[16] manufactured by the Humboldt company, with two meters diameter ($78\frac{3}{4}$ in.), has a capacity of 450 to 510 kg. (990 to 1122 ℔.) when making 26 r.p.m. (time not given) from a feed size of 30 to 50 mm. in diameter, crushed down to 3 mm. and less. The pulp contains 3 to 1.5 mm., 65 per cent.; 1.5 to 1 mm., 8 per cent.; 1 to 0.5 mm., 10 per cent.; 0.5 to 0.25 mm., 12 per cent.; .25 to 0.0 mm., 5 per cent.

Tube-Mills in the Transvaal.[17]—Report of the Government Mining Engineer deals with experiments at the New Goch and the Glen Deep mills, in which the battery has a coarse screen, the pulp is classified, sending the coarser part to a tube-mill using shaking amalgamated plates after the tube-mill. As a result it was found that, with about 14-mesh screen (200 per square inch), 6.5 tons per stamp per 24 hours were crushed, with about 12-mesh (150 to the square inch) 7.8 tons, with 10-mesh (100 to the square inch) 8.04 tons, and finally with 8-mesh (64 to the square inch) 10 tons were crushed. Of this last, 50 per cent. was made fine enough by the stamp. Ninety-four per cent. of the tube-mill product will go through a 60-mesh screen (3600 meshes per square inch). Mr. H. Leupold[18] gave results of tube-mill at Treasury gold mine, South Africa, for fine grinding. The stamps, weighing 1140 ℔., with 26-mesh screen, yielded 7 per cent. of concentrates averaging 12,7 dwt. gold and residue from cyaniding 2,13 dwt.; 59 per cent. of sand with 4.2 dwt. and residue 1.19 dwt.; 34 per cent. of slime averaging 2.9 dwt. and residue .76 dwt. A Krupp tube-mill 3 ft. 6 in. by 13 ft. was added, consuming 20 h.p., and with this the concentrates residue was eliminated entirely, while the sand residue fell from 1,19 dwt. to 1,10, and the slime residue from .76 to .61 dwt. The total added cost was 6.58c. per ton, and the saving was 30c. per ton.

Albert W. Swalm,[19] Consul, Southampton, England, writes that the

[14] *Mines and Minerals*, Vol. XXV (1904-5), p. 312.
[15] *Tonindustrie Zeitung*, Vol. XXIX (1905), p. 1384; *Oesterreichische Zeitschrift*, Vol. LIII (1905), p. 491.
[16] *Zeitschrift für Berg-, Hütten- und Salinenwesen*, Vol. LIII (1905), p. 125.
[17] *Mining Reporter*, Vol. LI (1905), p. 259.
[18] *Trans.* South African Association of Engineers, *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 1104; New Zealand *Mines Record*, Vol. VIII (1905), p. 429.
[19] Monthly Consular Reports, No. 296, May 1905, p. 82.

tube-mill in Transvaal has proved a great success. It increases the extraction of gold by 5 per cent., and the crushing power of mills can be augmented with less expenditure of capital than would be necessary for the purchase of the extra stamps.

At the Lake View Consols[20] there are three tube-mills 3 ft. 11 in. by 16 ft. 5 in. treating the product of stamps which has passed a 25-mesh screen. The capacity of each tube-mill is 67 tons of battery pulp per 24 hours, grinding to 220 mesh consuming 30 h.p. One set of liners wore out in 145 days, corresponding to 10,350 tons crude ore. The wear of the flint stones amounted to 848 lb. per 1000 tons of crude ore.

At the Ivanhoe Gold Corporation, Ltd., the sands fed into the tube-mill and the finished product had the following sizes:

	Sands fed.	Finished product.
On 40 Mesh.	38%	0.3%
Through 40 mesh on 60 mesh.	41.6	10.14
" 60 " " 100 "	15.4	39.63
" 100 " " 150 "	1.8	9.63
" 150	2.8	40.30

Alfred James[21] gives the regular tube-mill practice on the Rand. In grinding sand a 50 per cent. pulp gives good results; even 60 per cent. can be used. The most satisfactory rate of revolution is found to be $200 \div \sqrt{D}$ when D is the diameter of the mill in inches. The correct charge of flints is W=44xN, where W is the weight of flints and N the cubic foot capacity of the tube-mill cylinder. But West Australian wet grinding uses nearly 50 per cent. more.

The size of pebbles for wet grinding is 3 to 4 in. diam.; for dry grinding finer pebbles give finer product. It takes much less power to run a tube-mill wet than dry. For new installations, expected to grow, a larger tube-mill may be installed, using slower speed or less balls.

Arthur Lakes[22] calls attention to the chert of Lower Carboniferous limestone and the quartzite in Lower Cambrian and Algonkian of Colorado in case hard stones are required for tube-mills.

H. A. White[23] develops the theory of the tube-mill and draws conclusions to guide the millman. He deduces the theoretical path of the falling balls with various speeds of revolution. He found in his experiments with glass beads that higher speeds were necessary to follow these theoretical paths. He found in dry crushing that most of the pulverizing was done by the impact of the falling balls; he therefore advises in wet crushing to have the outlet so low that the water level will come but little higher than the top of the balls at the foot of the slope. In the discussion following he

[20] *Mines and Minerals*, Vol. XXV (1905), p. 300; Institution of Mining and Metallurgy, Advance Sheets.
[21] *Mining Reporter*, Vol. LII (1905), p. 135. *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 511, abstracted from Institution of Mining and Metallurgy, January 19, 1905.
[22] *Mines and Minerals*, Vol. XXVI (1905-6), p. 53.
[23] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 539, abstract of a paper read before the Chemical, Metallurgical and Mining Society of South Africa.

advises a lining composed of balls, and that the tubes be filled two-thirds full of balls, and gives a table of the theoretical revolutions necessary for diameters of the tube-mill from one in. to 9 ft. The fact was brought out that smooth steel lining flattens the balls by sliding and wears out the lining, while rough lining raises the balls or pebbles a little higher, causing reduction by impact instead of attrition, with an increase in capacity.

Grinding Pans.[24]—The success of the grinding pan over the tube-mill at Ivanhoe Gold Corporations mine, Kalgoorlie, Australia, under the auspices of a metallurgist in behalf of the tube-mill, in grinding coarse sands from the stamp-mill, is commented upon editorially. The tube-mill grinding from 15 mesh left 4.93 per cent. on 40-mesh screen while the pans left only 0.12 per cent. By returning the unground sand the pan gave 3.25 per cent. on 150-mesh while the tube-mill gave 4.00 per cent.

R. B. Nicholson and H. J. Brett[25] experimented at the Ivanhoe mill, Kalgoorlie, to determine the relative efficiency of two grinding-pans and one 13-ft. tube-mill. The two pans seemed to have the best of it as to first cost, running cost, capacity and efficiency in grinding.

The two pans ground 29.1 tons per day to the tube-mill's 26.3 tons. The pans received pulp of which 26.56 per cent. was finer than 150 mesh, and ground it so that 95.09 per cent. passed through that sieve; the tube-mill brought the pulp from 25.94 per cent. passing through the sieve to 95.62 per cent. passing through. The two pans cost \$1412.30, the tube-mill cost \$3082.71.

The two pans crushing 21.3 tons per day had a running cost of 54¾c. per ton; the tube-mill crushing 19.5 tons cost 61½c. per ton.

Regrinding of Gold Ores.[26]—J. A. Wauchope found from one battery with 640 ℔. stamps crushing from 1½ in. diam. through 40 mesh that the capacity was 3 tons per stamp per 24 hours. On the other hand, by crushing through a ¼-in. hole he increased the output to 10 tons per stamp. The stamp therefore consumes $\frac{7}{10}$ of its time in crushing from ¼ in. to 40 mesh. This pulp he then reduced to 40 mesh by a slow speed Lane edge roller with six rollers, making 10 r.p.m., making a cheap substitute for the expensive stamp. The total cost of crushing and concentrating was 62⅝c.

Limit of Crushing.[27]—E. E. Wann advocates both for amalgamation and cyaniding the grinding of every ore particle to the size of the most minute particle of gold contained.

Screening.

A valuable paper has been presented by Mr. Walter McDermott, chairman of the committee on standardization, of the Institution of Mining and

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24 *Mining Reporter*, Vol. LII (1905), p. 514.

25 *Mining Magizine*, Vol. XII (1905), p. 416, abstract from *London Mining Journal*, Vol. LXXVIII, Sept. 2, 1905; *Mining Reporter*, Vol. LII (1905), p. 284, abstract from Monthly Reports Chamber of Mines, West Australia.

26 *Mining Magazine*, Vol. XII (1905), p. 279.

27 *Mining and Scientific Press*, Vol. XCI (1905), p. 410.

Metallurgy, London, on the Standardization of Screens, asking for standards of sieves and sieve scales, and seeking contributions from men in every part of the world. Answers have come from the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa,[28] from Mr. W. Spencer Hutchinson[29] and from Mr. Courtenay DeKalb.[30] The writer here summarizes and comments upon these papers.

The first thing needed seems to be to state the want. The South African man needs to define the difference between sand and slime, sand being such material as can be treated by percolation and slime being that which requires agitation. Mr. Mumford suggests that 150-mesh screen represents the dividing line between these two for quartz, and 250-mesh the dividing line between them for pyrites. The California gold-mill man, who has adopted the Gates canvas plant, uses a rising current of 14 mm. per second to lift out his slimes that are to be concentrated further by the canvas table. The limiting size of this slime will be about .15 mm., all the larger part being rejected by the spigot of the classifier. The concentrator man working fine material (slimes) on tables has a variety of values. Linkenbach places the limit of slimes to be treated on a slime-table at .25 mm. The Central Lead Company, of Missouri, and the Boston and Montana Copper Company, of Montana, feed slimes to round tables which have from 5 to 10 per cent. coarser than .25 mm. The Bunker Hill and Sullivan Mining Company, of Idaho, feeds to its round tables slimes of which 5 per cent. is coarser than 0.3 mm. The Smuggler Mining Company feeds slimes to its round tables of which 5 per cent. is coarser than .125 mm.; the Lake Superior Copper slimes are of much the same size.

No other wants have been indicated by any of the contributors. It would seem to the writer that the concentrator men need sieve scales for mill concentration, and the laboratory man needs a sieve scale for valuing samples of ores.

In regard to the mill man's sieve scale, all the observations point to the fact that he is satisfactorily served at the present time, both as to the series of sieves and as to the apertures of the sieves. The determination of his series of sieves is done principally by mill work, modifying it from time to time in the purchase of new screens as his concentrating results indicate. The commercial names and numbers serve him perfectly for giving his orders, and his own accurate measure tells him when the hole is the size he desires.

Rittinger's sieve scale, advancing by geometric progression, has a very decided advantage for the mill man, for the reason that in sizes above 1 mm. it is true or nearly true that a ½ mm. grain of galena, for example, can

[28] Vol. VI (1906), p. 115.

[29] *Trans.* American Institute of Mining Engineers, Vol. XXXV (1905), p. 256; *Engineering and Mining Journal*, Vol. LXXX (1905), p. 213.

[30] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 151.

be settled away from a 1 mm. grain of quartz just about as easily as a 2 mm. grain of galena can be settled from a 4 mm. grain of quartz, or an 8 mm. grain of galena from a 16 mm. grain of quartz. This principle points directly to the geometric progression as the mill man's ideal sieve scale, and from it he naturally will depart only so far as his market for purchasing screens and the results of his mill practice seem to indicate.

When we come to the laboratory man we find here the greatest need for some definite agreement betwen experimenters and writers. The author has made many tests and written some papers in regard to the sizes of sands. He has tried to get a set of sieves that would conform to Rittinger's scale, with indifferent success. They none of them hit the exact figure; the holes in the cloth are not square. The requirements which seem to the writer to be absolutely essential are that the linear measure of the hole should be known, from which it may be assumed that we have the diameter of the grain which passes through. Secondly, that the sieves of the series should be sufficiently frequent; by that is meant that there should be no great gap in the series of sizes. The writer has taken the ground in favor of the Rittinger scale and of what he calls double Rittinger, that is to say, interpolating a size between each number of Rittinger's scale, because this is a simple series starting from a known point, namely, 1 mm. and going upwards and downwards by definite gradations.

The use to which the laboratory man puts this series of sieves is in the valuing of samples of sand from crushers, to see if the percentage is high in the coarse sizes, showing careful graded crushing, or high in the fine sizes, showing excessive sliming, and also to locate if there are any other peculiarities, such as an increase or a decrease, more than ordinary, at any part of the series of sizes. In order to do this satisfactorily, one must have a set of sieves, as stated above, of which the linear dimensions of the hole are known to him, and the series must be complete without any serious gaps in it. If a laboratory man in Australia reports the result of his crushing machines in terms of sizes and weights of grains resting on his series of sieves, the laboratory man in Montana can read this paper with satisfaction, and compare his own sizing tests on the crushing of Montana ore, making up his mind whether he is up to the standard of the Australian mill or better. He can, however, more speedily make his comparison if the system called the cumulative curve be adopted. From this he can read off directly the percentages of the sands that lie between 2 and 4 mm. in size or 2½ and 5 mm. in size, or any other pair of sizes, whether those sizes exist in the series with the Australian experimenter or not. The cumulative curve is in the nature of a universal language from which any one can read at a glance the whole story of the sizing test. The question has been raised as to whether millimeters or inches should be adopted. The writer argues for millimeters, because of

their ease in translation from weight to area and area to velocity in all classifier computations. He has had to compute many classifiers, and has found that the metric system lends itself with great ease to these computations.

The Ferraris Waving Screen.[31]—Charles Will Wright describes the Ferraris waving screen as used at Monteponi, Sardinia, and compares it with the revolving trommel. The Ferraris screen is an oblong sieve (12 ft. by 4 ft.) supported horizontally upon beechwood legs inclined at an angle of 65 deg. from the horizontal, and is given a reciprocating motion by an eccentric. This motion causes the ore to move rapidly forward by a series of jumps. At Monteponi three sieve sizes are used to each screen, the apertures of which are 14 mm., 20 mm. and 30 mm. in diameter. The capacity of this screen is 450 tons a day, while the capacity of three trommels (36 in. by 72 in. revolving at 20 r.p.m.) screening to these sizes is 150 tons per day. The trommels use 2 h.p. and take considerable time and expense to repair, while the Ferraris screen uses only 1 h.p. and the worn screens are replaced quickly and at a small cost. The trommel runs more smoothly, but the vibration due to the waving screen is insignificant in a mill operating stamps or rock-breakers.

The Imperial Ore Screen.[32]—A wet screen manufactured by John A. Traylor, Denver, Colorado, has a differential horizontal movement combined with a resilient vertical bump at the end of the stroke. The pulp is carried forward by the movement of the screen, and is suddenly tossed in the air by the vertical impingement of the screen on a stop at the end of the stroke, the screen quickly drawing back from under the pulp while it is still in the air, allowing the latter to drop back farther forward on the screen. The range of material which the screen is claimed to handle is from run of mine to 100 mesh. In comparison with the trommel, the makers put forth the following claims for this screen: less water necessary, less wear, higher capacity, no clogging of meshes, and ¼ head of a trommel required.

Pratt Ore-Sizer.[33]—A. H. Wethey describes a very satisfactory device, the Pratt ore-sizer, which has been installed at the Butte Reduction Works, for separating fine and coarse material leaving the fine jigs. This separation saves grinding the whole in the Chili mills; 47 per cent. of the material coming to the machine will pass 16-mesh screen. Each machine treats 150 tons per day, 60 tons of which pass through its screens. The machine consists of a cast-iron distributor with six discharges, which distribute the pulp by centrifugal force over a perforated stationary screen. The fine material passes through and thence to the concentrating tables, the coarse collects in basin at the bottom and goes through a

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[31] *Mining Magazine*, Vol. XI (1905), p. 333
[32] *Mining Reporter*, Vol. LI (1905), p. 10.
[33] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 435.

similar machine, the coarse material from the latter going to the Chili mills to be reground. The screens used are sheet steel, punched with 1.5 mm. round holes. They last three weeks and cause no trouble by blinding. The largest size particle coming to the machine is 4 mm.

CONCENTRATING MACHINERY.

Spitzlutte.[34]—H. Leupold offers as an improvement on the ordinary Bilharz spitzlutte a spitzlutte made by him for the Treasury mine. He built it of redwood treated with boiling linseed oil, which wears better than cast iron under action of sand and water. He gives the details of its construction, and also gives the following results as an average of many samples, both of concentration and overflow:

Meshes per Linear Inch.	Spigot Product.			Overflow.		
	Per cent.	Dwt. per ton	Gold value per cent.	Per cent.	Dwt. per ton	Gold value per cent.
Through 17 on 40.......	63.67	6.22	43.41	6.31	4.3	5.49
" 40 " 60.....	28.81	12.60	39.79	10.41	4.5	9.48
" 60 " 80.....				5.03	4.83	4.93
" 80	7.52	20.52	16.80	78.25	5.06	80.10

This table shows the spigot product to consist of 63 per cent. coarse sand of low value, a middling product and a product of 7½ per cent. fine pyrites of over 20 dwt. value. The overflow contains fine sand of 4½ dwt. and all the slime.

The Dillon Concentrator.[35]—This is a riffle table of the Wilfley type manufactured by the Dillon Iron Works Company. The concentrating surface consists of rubber corrugated lengthwise for the whole length of the table, and there are also riffles running the whole length of the surface.

The claim is made that the machine is very efficient in handling pulp containing a large proportion of fine mineral. The reasons given are: (1) that the fine mineral is stratified in the corrugations and travels along protected by the pulp on the top; (2) that the space between the groups of riffles gives the pulp a chance to stratify in smooth flowing water. The stroke can be varied from a smooth differential reciprocating motion to a hammer-like blow. Records extending over several months show an average extraction of over 90 per cent. in a concentrating mill equipped with these tables.

The Bartlett Simplex Concentrator.[36]—This new Bartlett table, manufactured by the Colorado Iron Works Company, is a machine with three rubber covered riffle decks mounted and vibrating on hickory strips. Its motion is of the quick forward and quick return type, and the pulp passes fastest over the first table and slowest over the last table, thus giving more time to

[34] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 1090, abstracted from the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, Vol. V (1905), p. 238.
[35] *Ibid.*, Vol. LXXIX (1905), p. 299.
[36] *Ibid.*, Vol. LXXX (1905), p. 121.

settle and stratify. The machine weighs 900 lb. and requires but $\frac{1}{6}$ actual h.p. to operate. The claim is made that this table saves slime so closely that no after treatment is required.

The Buss Slimer.[37]—Edward Walker describes the Buss slimer made by the Luhrig Company, London. It is essentially a canvas belt traveling in the direction of its length over rolls. The belt has a slope in the direction of its width. The whole belt is given an end knocking by a cam at the head end. It treats ore from 50 mesh down to the finest powder that will settle in water. This fine ore is fed at the highest side of the belt, the gangue is washed off at the lower side, and the concentrates are carried up with the belt and drop off over the end into a tank.

General Milling Practice.

Concentration of Copper Ores in the Southwest.[38]—Frank H. Probert deals with the Morenci and Clifton mills, in Graham county, Arizona. The author describes in detail the practice at the West Yankee, the Shannon and the Longfellow concentrators.

The West Yankee concentrator (Detroit Copper Mining Company) has a daily capacity of 650 tons of ore. The ore consists of a highly altered porphyry containing disseminated particles of chalcocite, and averages about 4 per cent. copper. Nearly 60 per cent. of the copper is taken out before recrushing, to prevent loss in sliming. In a comparative test between Bryan mills and high-speed rolls used in recrushing, the former produced less slimes and were repaired more quickly than the latter; they were therefore substituted for the latter. The following system of concentration is used:

1. Shaft. First-class ore, 7 per cent. and over, to smelter; second-class ore to (2).
2. Screens. Fine to (3); middle to (4); coarse to (5).
3. Briquetting machines at smelter.
4. Hand-picking belt. First class to smelter; remainder to (7).
5. Hand-picking belt. First class to smelter; remainder to (6).
6. Breakers. To (7).
7. Stock bins at concentrator. To (8).
8. Trommels. Oversize to (9); undersize to (10).
9. Roughing rolls. To (10).
10. Elevator. To (11).
11. Trommels (mm. not given). Coarse to (12); fine to (13).
12. Collom jigs.
13. Classifier. Spigot to (14); overflow to (15).
14. Jigs. Concentrates to smelter; tailings of all jigs to (16).

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[37] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 1106.
[38] *Ibid.*, Vol. LXXIX (1905), pp. 1088, 1224, and Vol. LXXX (1905), p. 15; *Mining Magazine*, Vol. XII (1905), p. 316, abstracted from *Trans.* of Seventh American Mining Congress.

15. Settling tanks. Slimes to (22); clear water returned.
16. Settler. Coarse to (18); middle to (17); fine to (19).
17. Settler. Coarse to (18); middle to (19); slime to (15).
18. Bryan mills, screen 1½ mm. To (19).
19. Hydraulic classifier. Coarse to (20); middlings to (21); slimes to (15).
20. Fine jigs. Concentrates to smelter; tailings to (23).
21. Upper vanners. Concentrates to smelter; tailings to settler to (22).
22. Lower vanner. Concentrates to smelter; tailings to (24).
23. Bartlett tables. Concentrates to smelter; tailings to (24).
24. Settler. Clear water returned; settlings to waste.

The Shannon concentrator, with a daily capacity of 500 tons, has a system similar to other mills in this district.

The Longfellow mill (Arizona Copper Company) has a daily capacity of 350 tons of an ore similar to that treated by the Yankee concentrator. The scheme of the concentration is as follows:

1. Stock bins. To (2).
2. Breaker, 1½ in. ring. Product to (3).
3. Roughing rolls. Product to (4).
4. Elevated to two trommels. Products sized ¼, ⅜, ½, and ¾ in. to (5).
5. Separate jigs for each size. Concentrates to smelter; tailings to (6).
6. Classifier. Coarse to (7); slimes to (12).
7. Huntington mills, 2 and 2½ mm. screens. Product to (8).
8. Hydraulic classifier. Slimes to (10); coarse product to (9).
9. Vanners (corrugated belt). Concentrates to smelter; tailings to (11).
10. Settling tanks. Settlings to (11); clear water returned.
11. Huntington mills, 1 and 1½ mm. screen. Product to (12).
12. Vanner (plain belt). Concentrates to smelter; tailings to (13).
13. Settling tanks. Settlings to waste; clear water returned.

The author further gives tabular results in some interesting sizing tests on the ground products of the Bryan mills, Huntington mills and high-speed rolls used at the above mills. He also gives tables of the percentage of copper found in the different products. He further states that the Elmore oil process was tried on large scale on the slimes containing 1.12 per cent. copper, but proved a failure. He gives cost of concentrating as 60c. per ton.

Concentration Methods at the Arizona Copper Company, Clifton, Arizona.[39] —A. Morrison describes briefly the five concentrating mills of the Arizona Copper Company. These plants have a total daily capacity of 1500 tons. Four run on sulphide ore with 4 per cent. copper, and produce concentrates of 18 to 20 per cent. copper. The fifth produces concentrates running 12 per cent. copper, and sends tailings running 2 per cent. copper to

[39] *Mining Reporter*, Vol. LII (1905), p. 8.

the sulphuric acid leacher, where the copper in the tailings is reduced to 0.25 per cent.

Concentrator No. 1 handles 325 tons daily of a 4 per cent. sulphide ore. The following system is used:

1. Blake breaker, 12x20 inches. To (2).
2. Cornish rolls, 18x36 inches. To (3).
3. Two trommels, diameter 36 in., length 14 ft. Four sections with $\frac{1}{4}$, $\frac{3}{8}$, $\frac{5}{8}$, and $\frac{3}{4}$ in. punched holes. Oversize to (4); undersize screen products to (5).
4. Cornish rolls, 14x22 inches. Product elevated to (3).
5. Eight Harz jigs. Concentrates to smelter; tailings to (6).
6. Two 5-foot Huntington mills, 3 mm. screens. Pulp, 3 per cent. copper, to (7).
7. Hancock jig. Concentrates, 16 per cent. copper, to smelter; tailings, 1.75 per cent. copper, to (8).
8. Classifiers in concentrator No. 4. Fines to (10), coarse to (9).
9. Two 5-foot Huntington mills, 1 mm. screen. Pulp to (10).
10. 36 Frue vanners. Concentrates, 20 per cent. copper; tailings, 0.8 per cent. copper, run to waste.

The system followed at the other mills is similar to that of concentrator No. 1, with the exception that the eight Harz jigs after the trommels have been replaced by one Hancock jig, the latter doing better work with less power and less water.

The Newhouse Mine and Mill.[40]—This article contains a description of the Newhouse Mine and Smelters mill in Copper Gulch, three miles from Frisco, Utah. The ore averages 4 per cent. copper, with $2 per ton in gold and silver. The character of ore is chalcopyrite, with some gray and native copper in a granite gangue. The ore is crushed at the mine to $1\frac{1}{2}$ in. and is transported to the mill, where the following system is used:

1. Bin. To automatic feeder, to conveyor, to elevator, to (2).
2. Screen (14 mm.). Undersize (4); oversize (3).
3. Coarse rolls. Product to (4).
4. Double screen (7 mm. and 3.5 mm.). Over 7 mm. to (5); on 3.5 mm. to (6); through 3.5 mm. to (7).
5. Jig No. 1. Concentrates into cars; tailings to (16).
6. Jig No. 2. Concentrates into cars; tailings to (16).
7. 10-mesh trommel (wire No. 20). Oversize to (8); undersize to (9).
8. Jigs No 3. Concentrates into cars; tailings to waste.
9. Two 10-mesh screens (wire No. 18). Oversize to (10); undersize to (11).
10. Jigs No. 4. Concentrates into cars; tailings to waste.

[40] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 57.

11. Calumet sizers. 1st pocket to (12); other pockets to (13); overflow to (14).

12. Jigs No. 5. Concentrates to cars; tailings to waste.

13. Wilfley tables. Concentrates to cars; tailings to waste.

14. Cone settler. Slimes to (13); overflow to (15).

15. Settling tanks. Settlings to slime tables; overflow, jig water.

16. Screen (3.5 mm.). Oversize to (17); undersize to (7).

17. Fine crushing rolls. Product to (7).

The mill is in two units and makes five jig sizes and five table sizes, besides the slime. The daily capacity is 600 tons, producing 100 tons of concentrates.

Treatment of Copper Ore at Moonta, South Australia.[41]—M. L. Gascuel describes very briefly the treatment of a 4.5 per cent. copper ore at Moonta, South Australia. The ore is hand-sorted at the mine into sterile, rich (16 to 20 per cent.) and medium (3 to 5 per cent.). The sterile is rejected, the rich is sent to the smelter, and the medium is concentrated at the mill. The machines used at the mills are: jaw-breakers, rolls, coarse and fine jigs, round buddles and Wilfley tables. The author describes in particular the Hancock jig, which originated at this mine. The concentration is 6 into 1, and the concentrates run 15 to 17 per cent. copper; the tailings run 0.8 per cent. copper. The tailings from the Wilfley, running 2 per cent. copper, are leached with sulphuric acid.

The Treatment of Copper Rock at the Quincy Mills, Hubbell, Michigan.[42]—C. K. Hitchcock, Jr., gives a description, with flow sheet of No. 2 mill of the Quincy Mine, Houghton county, Michigan. The ore treated is a moderately hard amygdaloid, containing 1¼ per cent. native copper and varying amounts of quartz, calcite, epidote, and zeolites. The ore is crushed to 4-in. size at the mine, and is brought in railroad cars to the bins at the mill, where the following system is used:

1. Ore bins. To (2).

2. Steam stamps. Hydraulic mortar discharge to (3); through screen ⅝-in. holes to (4).

3. No. 1 grade copper. To smelter.

4. Hydraulic classifier. Spigot product to (3); remainder to (5).

5. Trommels (¼-in. holes). Undersize to 6; oversize by bucket elevator to (2).

6. Hydraulic classifier. Spigot product to (3); remainder to (7).

7. Roughing jigs. Heads to smelter; middlings to (8); hutch to (10); tailings to waste.

8. Chili mill (10-mesh screen). Pulp to (9).

[41] *Annales des Mines* Series 10, Vol. VII (1905), p. 55.
[42] *School of Mines Quarterly*, Vol XXVI (1904-5), p. 341.

9. Hydraulic classifiers. Spigot, No. 2 mineral to smelter; overflow to (10).

10. Finishing jigs. Hutch 1st and 2d jig, No. 3 mineral; hutch last jig to slime table; overflow to a special Wilfley table.

The stamp makes 100 strokes per minute at a steam pressure of 120 lb.

Pyrite Mining in St. Lawrence County, New York.[43]—Robert B. Brinsmade describes the concentration of pyrite at the Cole mine, owned by the Adirondack Pyrite Company. The ore, which consists of a pyrite in a quartz and feldspar gangue, is brought in mine skips to the mill, where the following system is used:

1. Blake breaker (14x18 in.) crushing wet. To (2).
2. Cornish rolls (12x24 in.) crushing wet. To (3).
3. Elevator. To (4).
4. Trommel ($\frac{3}{16}$ in. holes). Oversize to (5); undersize to (6).
5. Rolls (14x19 in.). Product elevated to (4).
6. 4 Harz jigs, 3 compartment, 24x32 in. 4-mesh No. 16 wire. Screen concentrates to (7); hutch to (8); tailings to (9).
7. Elevator to storage bin, to car.
8. Elevator to hutch storage bin, to arc.
9. Centrifugal sand pump, to flume, to tailings heap.

The raw ore runs 20 to 30 per cent. sulphur, the concentrates run 40 to 50 per cent. sulphur, and the tailings run over 5 per cent. sulphur. The cost of milling is $0.37 per ton of raw ore or $0.74 per ton of concentrates. Capacity 55 tons per 10 hour shift.

Pyrite Mining in Virginia.[44]—R. H. Painter describes the mining of pyrite and also briefly the milling of pyrite at three mills in Virginia. The ore is pyrite, carrying a small amount of copper enclosed in a country-rock of slate and schist. The ore is separated into lump, spall, and fine. At the mill of the Louisa mine, which is typical of all the mills of this district, the ore is crushed successively by breakers and rolls, screened by trommels and distributed to Harz jigs. Jigging through a bed of cast-iron balls is employed, and the hutches are elevated by a belt-conveyor to the stock piles. Other mills differ from this mill in that they employ hand sorting and spalling on clean lumps of pyrite.

The author advocates departing from the usual method of jigging sized products by themselves and adopting the method of combining fine and coarse on the same jig to make both discharge and hutch according to some specified law; for example, with six sizes he would combine 1 and 4, 2 and 5, and 3 and 6, to feed them to the jigs. The advantages claimed are half the number of jigs, half the floor space, half the power of jigging, no clogging of screens and a good lively bed.

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[43] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 770.
[44] *Ibid.*, Vol. LXXX (1905), pp. 148 and 433.

Pyrite Deposits of the Western Adirondacks, New York.[45]—Edwin C. Eckel describes briefly the milling of pyrite at the Stella mine, St. Lawrence county, New York. The ore as it comes to the mill contains 26 to 27 per cent. sulphur. By concentration the sulphur content is raised to 44 to 45 per cent. Clean lump ore is shipped if it contains 34 to 35 per cent. sulphur. The mill practice is as follows: After picking out the high grade lump ore the remainder is broken to 1½ in., then by rolls it is further reduced to ¼ in. The material then passes through two sets of screens (¼ in. and ⅛ in.), the products going to two different sets of jigs. The jigs were Harz, but at time of writing they were being replaced by New Century jigs.

Washing Brown Hematite Ores.[46]—W. R. Crane describes the iron ore washer of the Tennessee Coal, Iron and Railroad Company at East Giles, 35 miles southwest of Birmingham, Alabama. The ore is a soft brown hematite mixed with clay and some sand; the iron content varies. The ore is loaded into cars by steam shovels and brought to the mill, where the following system is used:

1. Receiving bin, from which ore is washed by a hydraulic jet along a sloping floor into (2).
2. Grizzly (30 lb. rails, spaces 6 in.). Undersize to (3); oversize to (4).
3. Semi-cylindrical sheet steel chute. To (4).
4. Feed-hopper. To (5).
5. Wash trommel (3 and 4 in. holes). Undersize to (6); oversize to (12).
6. Receiving hopper. By launders to (7).
7. Two double log-washers. Discharge to (8); overflow to (10).
8. Two double trommels (mesh ⅛ in.). Undersize to (9); oversize to (11).
9. Receiving hopper. By launders to (10).
10. Waste bank.
11. Finished product. To railroad.
12. Rubber picking belt. Clay balls and stones to (10); discharge to (13).
13. Grizzly (1 in. bars spaced 4 in.). Undersize to (10); oversize to (11).

The iron content of the washed ore ranges between 35 and 55 per cent. The capacity of the mill is about 600 tons per day.

The Concentration and Separation of Zinc-Lead Ores in Summit County, Colorado.[47]—David H. Lawrance gives description of the practice at the old Union Mill, Breckenridge, Colorado. The ore is a zinc-lead-iron ore containing gold and silver values. The system used at the mill is as follows:

1. Breaker. Product to (2).
2. Coarse rolls. Product to (3).

[45] Edwin C. Eckel, *Bulletin* No. 260, United States Geological Survey, p. 587.
[46] *Mines and Minerals*, Vol. XXV (1905), p. 417.
[47] *Mining and Scientific Press*, Vol. XCI (1905), p. 365; *Mining World*, Vol. XXIII (1905), p. 557; *Mining Reporter*, Vol. LII (1905), p. 452

3. Screen (8 mesh). Oversize to (4); undersize to (5).

4. Fine rolls. Product to (3).

5. Screen (10 mesh). Oversize to (6); undersize to (7).

6. Jigs 1 and 2. Concentrates iron and lead in different compartments; tailings zinc and silica to (9).

7. Screen (24 mesh). Oversize to (8); undersize to (10).

8. Jigs 3 and 4. Concentrates same as (6); tailings same as (6) go to (9).

9. Jigs 7 and 8. Concentrates zinc; tailings to (18).

10. Screen (30 mesh). Oversize to (11); undersize to (12).

11. Jig 5, six compartment. Separates zinc, iron and lead in one operation; tailings to (18).

12. Hydraulic classifiers. Coarse to (13); medium to (14); fine to (16).

13. Jig 6. Products and jig same as (11).

14. Wilfley tables 1, 2, 3, and 4. Concentrates; overflow to (15).

15. Wilfley tables 7 and 8. Zinc concentrates; tailings to (18).

16. Wilfley tables 5 and 6. Concentrates; overflow to (17).

17. Wilfley slimers 1 and 2. Zinc concentrates; overflow to (18).

18. Tailings sluice to waste.

The mill ships three products: (1) The lead concentrates carrying high silver values and some gold; (2) the iron concentrates carrying gold and silver values; (3) the zinc concentrates carrying gold and silver values. The capacity of the plant is 120 tons per day. The concentration is 4 into 1.

Lead and Zinc Concentration at Maiern bei Sterzing.[48]—Mr. Rose gives a description of the concentrating mill at Maiern. The ore is a mixture of galena, blende and spathic iron with gold and silver values. The scheme of treatment of the ore is as follows:

1. Hand-picking tables at Schneeberg. Blende to (2); galena to (3); ore knockings to (4); blende middlings to (7); mine fines under 60 mm. to (5); waste to dump.

2. Lead smelter at Przibram, zinc smelter in Steiermark.

3. Ore-dressing plant at Seemos.

4. Rock-breaker at Maiern. Product to (6).

5. Screens, 60 to 45 mm. to (6); 45 to 16 mm. to (7); under 16 mm. to (9).

6. Picking tables. Picked ore to (2); blende middlings (13); galena middlings collected and jigged; waste to dump.

7. Trommel (25, 22, 19 and 16 mm.). Products by elevator to (8).

8. Three coarse jigs. Blende to (2); blende middlings to (6); tailings waste.

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[48]*Zeitschrift für Berg-, Hütten-, und Salinenwesen*, Vol. LIII (1905), p. 189.

9. Trommel (13, 11, 9, 7, and 5 mm.). Over 5 mm. to (10); under 5 mm. to (11).

10. Elevator to five fine jigs (16, 13, 11, 9, 7, mm. products). Blende to (2); blende middlings to (13); galena middlings collected; tailings waste.

11. Trommel (4, 3 and 2 mm.). Over 2 mm. to (10); under 2 mm. to (12).

12. Two fine jigs. Galena middlings collected and jigged; blende concentrates to (2); blende middlings to (12); tailings to waste.

13. Five roasting furnaces. Product through ore sampler to (14).

14. Two preliminary rolls. To (15).

15. Two screens (3 mm.). Over 3 mm. to (16); under to (17).

16. Elevator to two screens (10 mm.). Under 10 mm. to (17); over 10 mm.

17. Elevator to two fine rolls. To (18).

18. Two screens (3, 2, 1, ½ mm.). Products to (19).

19. Two magnetic separators, Nos. 1 and 2. Blende-bearing spathic iron; sizes 2 and 3 mm. to (20); size 1 mm. to (23); size ½ mm. to (24); impure blende (3 to ½ mm.) to (25).

20. Elevator to fine rolls (1 mm. No. 3). To (21).

21. Screens (1 and ½ mm.). Products to (22).

22. Magnetic separator No. 3. Spathic iron to waste heap; impure blende (1 and ½ mm.) to (25).

23. Magnetic separator No. 5. Impure blende (1 mm.) to (26); spathic iron (1 and ½ mm.) to waste; impure blende (½ mm.) to (25).

24. Magnetic separator No. 4. Products same as (23).

25. Spitzlutte. Over ½ mm. to (26); under ½ mm. to (29).

26. Four fine jigs, 4 compartment, 3, 2, 1, ½ mm. Blende concentrates to (2); blende middlings (3½ mm.) to (27); galena middlings (½ mm.) to (28); tailings to waste.

27. Four repeating jigs (3 to ½ mm.). Blende concentrates to (2); blende middlings to (27); galena concentrates to smelter (2); tailings to waste.

28. One repeating jig (½ mm.). Galena concentrates to (2); blende concentrates to (2); blende middlings to (27); tailings to waste.

29. Spitzkasten (4 compartments). Sands to (30); slimes to (31).

30. Two bumping tables. Galena to (2); blende to (2); middlings to (30); tailings to waste.

31. Two bumping tables. Galena to (2); blende to (2); middlings to (31); tailings to waste.

The different zinc concentrates run anywhere from 40 to 57 per cent. zinc. The lead concentrates run 62 to 68 per cent. lead and carry the gold and silver values. The concentration is about three into one.

Concentration of Silver-Lead Ores in Australia.[49]—V. F. Stanley Low describes the concentration of silver-lead ores at Broken Hill, New South Wales, in general, and more particularly the practice at the new mill of Block 10 mine. The mill which the author describes has a capacity of 3600 tons, and is in four divisions independently driven by motors. The system used at the mills is as follows:

1. Ore broken to 10 in. by 8 in. in mine. To (2).
2. Grizzly (1½ in.). Oversize to (3); undersize to (4).
3. Gates breaker. Product (1½ in.) to (4).
4. Storage bins at top of mill building. To (5).
5. Conical trommels (⅛ in. punched hole). Oversize to (6); undersize to (8).
6. Cornish rolls. To (7).
7. Trommels (2.5 mm. round holes). Oversize to (6); undersize to (8).
8. Classifier. Coarse to (9); fine to (13).
9. Coarse jigs. Hutches Nos. 1 and 2 to smelter; Nos. 3 and 4 to (10); No. 5 to tailings dump.
10. Krupp ball-mills. To (11).
11. Classifier. Coarse to (12); fine to (13).
12. Fine jigs. Hutches 1 and 2 to smelter; 3 and 4 to (10); 5 to zinc middling dump.
13. Spitzkasten. Coarse to (14); fine to (15).
14. Wilfley and Krupp tables.
15. Classifier. Products to belt vanners.

Trommel (5) is 6 ft. long, diameter 3 ft. large end and 2 ft. small end, covered with 14-gage iron punched with ⅛ inch round holes.

Rolls (6) are Cornish type gear-driven (7 to 1), shells 2 ft. 6 in. external diameter, coned centers bolted together. Each pair consists of a plain and a flanged roll. Run at 15 r.p.m., capacity 1000 tons per week from 1½ in. to ⅛ in., use 25 h.p.

Trommels (7) 6 ft. in length, 22 in. diameter. Make 20 r.p.m., inclination 1 in. to 1 ft., screens 14-gage iron, punched round holes 2½ mm. diam.

Classifier (8) is inverted cone 2 ft. diam. at base and 2 ft. 6 in. deep; it has a 2⅝-in. water inlet hole and one discharge hole at apex.

Jigs (9) are May Brothers' patent, have 8 working and 2 tailings compartments, 5 on a side. Each working compartment is separate with plunger at bottom; each hutch is 3 ft. 6 in. by 2 ft. 6 in. Plunger is same dimensions, with clack opening 2 ft. 6 in. by 6 in. Screen is 6-mesh brass wire. Each plunger makes 180 pulsations per minute. Each jig requires 2 h.p. and treats 6 to 7 tons per hour.

[49] *Trans.* of the Australasian Institute of Mining Engineers, Vol. X (1905), p. 197.

Krupp ball-mills (10) crush to 20-25 mesh, run at 30 r.p.m. and use 8 to 10 h.p.

Fine jigs (12) are similar to coarse jigs, except that they have smaller hutches and plungers, and have 200 strokes per minute and use 1½ h.p. per jig and have a capacity of 4.5 tons per hour. Wilfley (14) runs at 220 to 240 vibrations per minute and has capacity of 1 ton per hour for 30-mesh material and of ½ ton per hour for material from 30-mesh down to slimes. Three-quarter h.p. per table is used. The 6-ft. Chili mills run at 30 r. p. m. and have a capacity of 100 tons per 24 hours. Water is first added to ore in these mills. This large capacity is due to softness of gangue. In the jigger-room the lead concentrates are freed from their water by being jigged up and down in a wheelbarrow on a platform jarred by a revolving cam. The following table shows approximate values obtained in different concentrates:

	Wilfley Concentrates	Blake Products.	
		Lead.	Zinc.
Gold	0.20 oz.	0.10 oz.	....
Silver	24.00	16.00	....
Lead	50.00%	15.00%	....
Zinc	7.00	9.00	48 to 50%
Iron	12.00	30.00	....
Silica	3.00	3.00	....

Jackson Concentrating Works, Idaho Springs, Colorado.[50]—C. M. Rath and R. L. Grider describe the Jackson concentrator at Idaho Springs, Colorado. The ore is a mineralized quartz porphyry, with varying amounts of sulphides of lead, zinc, iron and a little copper. The treatment of the ore is as follows:

Ore is elevated to crushing-floor and goes to (1).

1. Blake breaker (7x9 in.). Wet crushed product 1¾ in. to (2).
2. Rolls (10x24 in.). Product maximum size ¾ in. to (3).
3. Trommel (3 mesh). Oversize to (4); undersize to (5).
4. Rolls (10x20 in.) Product less than ¼ in. to (3).
5. Trommels (5, 8, and 14 mesh). Oversizes to (6); through 14 mesh to (9).
6. Coarse jigs (Nos. 1, 2, and 3). Hutch to smelter; heads to (7).
7. Huntington mills (30 mesh). To (8).
8. Trommel (14 mesh). Oversize to jig No. 3 (6); undersize to (9).
9. Jigs (Nos. 4, 5, and 6). Concentrate to smelter; heads to (10).
10. Pulp divided into three portions. First to (11); second to (13); third to (15).
11. Spiral plane table No. 2. Concentrates to (19); middlings to (12); tailings to (18).
12. Small spiral plane table. Concentrates to (19).

[50] *Mining Reporter*, Vol. LII (1905), p. 547.

13. Spitzkasten. Coarse to (14); fine to (17).
14. Cammett table. Concentrates to (19); tailings to (22).
15. Weller classifier. Coarse to (16); fine to (17).
16. Spiral plane table No. 1. Concentrates to (19); tailings to (21).
17. Roberts vanner No. 1. Concentrates to (19); middlings to (18); tailings to (20).
18. Buddle No. 1. Concentrates to (19); tailings to (20).
19. Settling tank, to elevator, to concentrates bin, to smelter.
20. Settling tank. Overflow to sump.
21. Buddle No. 2. Concentrates to (19); tailings to (20).
22. Buddle No. 3. Concentrates to (19); tailings to (20).

The Blake breakers have 244 crushing strokes per minute. Rolls (2) make 70 r.p.m. Rolls (4) make 120 r.p.m. The Harz jigs have 160 impulses per minute; stroke in coarse jigs is $\frac{3}{4}$ in. and in fine jigs $\frac{3}{8}$ in. Mill has daily capacity of 50 tons. Two tons of coal per day at $5 per ton are used when boiler carries full load.

Plant for the Handling and Treatment of Ores at the Silver Cup and Nettie L. Mines, British Columbia[51].—George Attwood describes the treatment of the ore coming from these mines. The ore consists of galena, zinc, blende, gray copper with gold and silver values in a graphitic clay-slate gangue. The following system is used at the mill:

1. Ore coming by aerial tramway is dumped on (2).
2. Grizzlies (spaces 2 in.). Oversize to (3); undersize to (4).
3. Blake breakers (10x16 in.). To (4).
4. Four swinging automatic feeders. To (5).
5. Four gravity stamp-batteries. To (6).
6. Automatic sampler. To (7).
7. Classifiers. Sands to (8); slimes to (9).
8. Dodd buddles. Galena concentrates to smelter; middlings to (11).
9. Pointed boxes. Slimes to (10); excess water.
10. Frue vanners. Concentrates to (11); tailings waste.
11. Howell-White chloridizing furnaces. Product to (12).
12. Cold water. Leached copper to (13); residue to (14).
13. Scrap iron. Product copper to smelter.
14. Pans. Gold and silver amalgam to (15); tailings to waste.
15. Retort. Gold and silver bullion to market; clean mercury.

The stamps are of the heavy type, each stamp weighing 1000 lb. Each stamp crushes 4 to 8 tons in 24 hours, according to fineness of screen. Each Dodd buddle has a capacity of from 25 to 45 tons a day, according to percentage of metallics in the ore. The total cost of mining and milling is $7.83 per ton. Eighty to 82 per cent. of the lead is saved and 75 per cent. of the gold and silver is extracted. Power is furnished by four Pelton

[51]Minutes of Proceedings of the Institution of Civil Engineers, Vol. CLIX (1905), p. 295.

water-wheels working under low head. Less than 200 h.p. is used.

Classification as Applied to the Concentration of Finely Crushed Ores.[52]—J. M. Callow describes his method of classifying finely crushed ores as follows:

1. Crushing machinery ratio of ore to water=10:1. To (2).
2. Sloughing off tanks. Coarse (ore to water=4.6 to 1) to (3). Overflow to 11.
3. Traveling belt screen (20 mesh). Oversize to (4); undersize to (7).
4. Jigs (ore to water=12:1). Concentrates; tailings to (5).
5. Dewatering screen. Tailings to waste; clear water to (6).
6. Sump to pump, to tank at head of mill.
7. Traveling-belt screen (40 mesh). Oversize to (8); undersize to (9).
8. Wilfley tables. Concentrates; middlings to (8); tailings to waste; clear water to (6).
9. Traveling-belt screen (80 mesh). Oversize to (10); under to (11).
10. Wilfley tables. Concentrates; middlings to (10); tailings to waste; clear water to (6).
11. Callow tanks. Clear water to (6); pulp to (12).
12. Wilfley tables. Concentrates; middlings to (12); tailings to waste; overflow to (13).
13. Callow tanks. Clear water to (6); pulp to (14).
14. Wilfley tables. Concentrates; tailings to waste; overflow to (15).
15. Callow tanks. Clear water to (6); pulp to (16).
16. N. C. Slimers. Concentrates; tailings to waste.

By this system Mr. Callow claims to have advanced the practical limit of sizing from 2 mm. to 80 mesh or finer.

Notes on Concentrating Mills.[53]—Alfred Harvey gives notes and drawing of a 150-ton section of a mill. These notes describe no particular mill, but are taken from mills in active operation with which the author has been associated. The ore to be treated in this mill contains argentiferous galena, iron oxide and pyrite, some chalcopyrite and a little zinc. The gangue is chiefly quartz and slate. The iron ore is the main gold carrier. The mill system is as follows:

1. Ore passes over (2).
2. Grizzly (1¼ in. ring). Oversize to (3); undersize to (7).
3. Gyratory breaker. To (4).
4. Magnet and picking out nails, pick points, pieces of drills. Ore to (5).
5. Revolving screen (1¼ in. holes). Oversize to (6); undersize to (7).

[52] *Mining and Scientific Press*, Vol. XCI (1905), p. 449.
[53] *Mines and Minerals*, Vol. XXV (1905), p. 448

6. Two small gyratory breakers. To (7).
7. Belt conveyor. To (8).
8. Sampler No. 1. Sample $\frac{1}{10}$ to (9); rejected ore to (12).
9. Rolls (27x24 in.). To (10).
10. Sampler No. 2. Sample $\frac{1}{10}$ to (11); rejected ore to (12).
11. Sample bin where at end of day the whole sample is elevated and rerun through sampler No. 2 until cut down to 300 lb. Split shovel to 150 lb., then to sample grinders.
12. Storage bin. To shaking feed to (13).
13. Trommel (60x24 in.). (2 mesh, opening 0.308 in.) Oversize to (14); undersize to (15).
14. Rolls (34x16 in.). To (15).
15. Elevator to trommel (2 mesh, opening 0.308 in.). Oversize to (16); undersize to (17).
16. Rolls (27x14 in.). To (15).
17. Trommel (4 mesh, opening 0.145 in.). Oversize to (18); undersize to (19).
18. Two jigs. Concentrates; tailings to (28).
19. Trommel (8 mesh, opening 0.067 in.). Oversize to (20); undersize to (21).
20. Two jigs. Concentrates to market; tailings to (28).
21. Trommel (12 mesh, openings 0.042 in.). Oversize to (22); undersize to (23).
22. Two concentrating tables. Concentrates to market; middlings to (28).
23. Hydraulic sizer, three compartment. Two coarse sizes to (24); third size to (25).
24. Two concentrating tables. Concentrates to market; middlings to (28).
25. Spitzkasten, 4 compartments. First three products to (26); last product and overflow to (27).
26. Six concentrating tables.
27. Three slimers.
28. Three roller mills. To (29).
29. Copper amalgamating plates. Amalgam; pulp to (25).

Such a section of a mill will have a capacity of 175 tons per 24 hours.

Tin Mining in Bolivia.[14]—D. H. Bradley, Jr., gives a brief description of the dressing of tin ores in the Huanuni and Oruro districts, Bolivia. The author states that 1000 to 1200-lb. stamps are used to crush the hand-sorted ore, which runs 4 to 8 per cent. metallic tin. The mortar screens are 6 to 16 mesh, according to the ore. The pulp from the stamp after being sized in a trommel goes to jigs and Cornish round buddles.

[14] *Mining Magazine*, Vol. XI (1905), p. 41.

The heads are saved, the middlings mixed with a new charge and the tailings run into another buddle; this is repeated until every particle of ore has had five or six opportunities of being caught. Wilfleys are now being introduced, with a better product and greater saving.

Tin Mining in Indo-China.[55]—A brief description is given of the saving of the tin on the Tinh-Tuc grant, Indo-China. The alluvium-bearing tin is run over sluices of the Malacca type. The loss of fine ore is large. One cubic meter of alluvium yields 5 kg. of cassiterite. The concentrates average about 50 per cent. tin.

Mount Bischoff Tin Mine.[56]—Sidney Fawns describes the dressing of tin ore at the Mount Bischoff tin mine. See The Mineral Industry, 1904, p. 405.

The Treatment of Tin-Wolfram-Copper Ore at the Clitters United Mines, East Cornwall, England.[57]—The following scheme is in use at the mill:

After a preliminary crushing a small portion of the ore goes to (1); the remainder to (4).

1. Roll, 12x28 in., friction driven. To (2).
2. Vibrator screen. Fines to (3).
3. Ball-mill (30 mesh). Pulp to (5).
4. Four stamp-batteries. To (5).
5. Three Spitzlutten. Coarse products to (6); overflow to (7).
6. Separate Buss swinging tables. Heads to (11); middlings to (6); tailings to (10).
7. Condensing and classifying spitzkasten. Coarse products to (8); overflow to (10).
8. Four double Luhrig vanners. Concentrates to (9); tailings to (10).
9. Cornish buddle. Concentrates to (11); tailings to waste.
10. Spitzkasten; lime added to settle. Water pumped back through the batteries.
11. Calciner. Product to (12).
12. Wetherill magnetic separator. First field magnetic iron oxide waste; second field oxides of iron with adhering copper; third field wolframite with some oxides of iron; fourth field clean wolframite; non-magnetic tin oxide and silicious waste.

The stamp-batteries are of five stamps each; the weight of each stamp is 880 lb. The number of drops per minute is 95.

Reduction of Cinnabar at Sulphur Creek, California.[58]—The usual practice of treating ores of mercury is to divide the ore by rough screening over grizzlies, into coarse and fine, and to treat these two classes in shaft furnaces. At the Manzanita mine, Sulphur Creek, California, the ore was

[55] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 829.
[56] *Trans.* of the Institution of Mining and Metallurgy, Vol. XIV (1904-5), p. 221.
[57] *Proceedings* of the Institution of Mining and Metallurgy, Oct. 15, 1905; *Mines and Minerals*, Vol. XXVI (1905-6) p. 224.
[58] *Mining and Scientific Press*, Vol. XC (1905) p. 335

first worked for gold, but later the gold diminished in amount and the cinnabar increased, so it was decided to turn the gold mill into a cinnabar concentrating plant. The ore is a highly silicious metamorphic sandstone, impregnated with cinnabar, iron sulphide and other minerals associated with quicksilver. The method of treatment at the mill is as follows:

1. Grizzly. Oversize waste to dump; undersize to (2).
2. Gates breaker, crushing to less than 1½ in. Product to (3).
3. Huntington mill, grinding through 30 mesh. Product to (4).
4. Two Gilpin county bumping-tables. Concentrates to (6); tailings to (5).
5. Two bumping-tables. Concentrates to (6); tailings to waste.
6. Horizontal retorts. Vapor to (7); residue waste.
7. Condenser. Product is mercury to market.

The Huntington mill has a capacity of from 50 to 120 tons, according to the work. The concentrates from the first set of bumping-tables run 60 per cent. cinnabar. The second set of tables are set to save the fine cinnabar not caught in the first set.

Asbestos in Canada[59].—Fritz Cirkel describes the mining and milling of asbestos in Canada. Some of the mines send the entire output to the mill, others make a rough separation as follows:

1. Material from the pit containing long fiber to (2).
2. Hand sorting. Fiber to market; rock containing long fiber to (4); rock containing small fiber to mill; refuse and fines to (3); rock to dump.
3. Dryer. To mill.
4. Cobbing shed(men). Small stones with long fiber to (5); stone with small fiber to mill; refuse and fines to (3); rock to dump.
5. Screen $\frac{3}{16}$ in. Oversize to (6); undersize to (3).
6. Cobbing shed (girls). Fiber I to (7); fiber II to (9); refuse and fines to (3.)
7. Screen $\frac{9}{16}$ in. Oversize No. I fiber to (8); undersize to (9).
8. Bags. To market.
9. Screen $\frac{3}{8}$ in. Oversize No. II fiber to (8); undersize to (3).

The author gives four mill systems used; they differ only in minor details. The following is one of these systems:

1. Ore bin. To (2).
2. Blake breaker. Product to (3).
3. Rotary dryer, 30 ft. long, 3 ft. diam. Product elevated to (4).
4. Rotary crusher. Product to (5).
5. Fiberizer, 11 ft. long, 3 ft. diameter. Product elevated to (6).
6. Screen ($\frac{1}{16}$ in. holes). Undersize to (7); oversize to (8); fiber drawn by fan to (10).

[59] *Bulletin of Mines*, Canada; *Engineering and Mining Journal*, Vol. LXXX (1905), p. 924; *Canadian Mining Review*, Vol. XIV (1905), p. 151.

7. Sand dump.
8. Cyclone beater. Product to (9).
9. Shaking screen ($\frac{1}{16}$ in.). Undersize to (7), oversize to (8), fiber drawn by fan to (10).
10. Collector I. Material to (11).
11. Revolving screen (5-16 in. holes). Oversize grade No. 1 to (12); undersize grade No. 2 to (13).
12. Shaking screen (10 mesh). Fiber taken up by fan to (15); undersize to (14).
13. Shaking screen (12) mesh. Fiber taken up by fan to (17); undersize to (14).
14. Shaking screen (12 mesh). Fiber taken up by fan to (17); undersize to (18).
15. Collector II. Material to (16).
16. Shaking screen. No. 1 fiber to market; tailings to (18).
17. Collector III. No. 2 fiber or paper stock to market.
18. Emery mill. Finishing powder to market.

The fiberizer is a horizontal stationary cylinder through the axis of which passes a shaft carrying groups of four perpendicular arms, the groups separated by 6 in. The shaft revolves at 500 to 700 r. p. m. The material is fed at the top at both ends and discharges at the middle at the bottom. The blades are set so as to beat the material towards the middle. The Cyclone beater consists of a pair of chilled steel propellers facing each other near the bottom of a cast-iron chamber, and revolving at 2000 to 2500 r.p.m. in opposite directions. A propeller wears about two weeks. The feed is walnut size and the discharge is pea size; the capacity 40 tons per hour.

Average results per day at one mine are: 125 tons mined at cost of $0.53 per ton, 80 tons milled at cost of $0.80 per ton, finished asbestos 7.5 tons at total cost of $17.41 per ton. This asbestos brings all the way from $20 to $200 per ton, according to length of fiber and grade of material.

Mica and the Mica Industry[60].—George Wetmore Colles describes the mining and dressing of mica in India, Canada and the United States. The method of dressing in use in Canada and the United States is as follows: The plates of mica are split into sheets $\frac{1}{8}$ to $\frac{3}{16}$ in. in thickness. They are then washed to free them from dirt, graded in size and cut into rectangles of standard sizes and shipped to the market. In some mills, where the mica is to be used for insulating, it is sent to splitting-tables, where the plates are further reduced to less than 0.01 inch in thickness. The scrap mica is ground up into mica flour and sized to different degrees of fineness. Mica flour finds extensive use as filling for paper, plaster and lubricants, also for decorating wall paper.

[60] *Journal* of the Franklin Institute, Vol. CLX (1905), p. 327.

Mica, its Occurrence, Exploitation and Uses[61].—Fritz Cirkel gives a description of the mining and dressing of mica in Canada. The following system is used in dressing the mica: The run of the mine is thoroughly cleaned of the adhering rock matter; the crystals or fragments which are not wrinkled are split up into plates about $\frac{1}{16}$ in. thick by means of a short-handled knife sharpened on both edges at the point. The rough edges are thumb-trimmed and graded. It is packed in barrels and shipped to the trimming works. Here the mica is cleaned, thoroughly re-sorted, split and trimmed by machine knives. The scrap mica is ground in tube-mills or by buhrstones into a fine powder, the standard sizes of which are 20, 40, 60, 70, 80, 100, 140, 160 and 200 mesh.

Tennessee Phosphates[62].—R. D. O. Johnson describes the mining of phosphates in Tennessee in general and also describes in particular the mining and milling of phosphates by the New York and St. Louis Mining and Manufacturing Company. The scheme of treatment used at the above mill is:

1. Storage bins. Hand fed to (2).
2. Sturtevant rotary breaker to $\frac{1}{2}$ inch. Elevated to (3).
3. Two screens ($\frac{1}{4}$ in. and 60 mesh). Coarse crushed ($\frac{1}{4}$ to $\frac{1}{2}$ in., to (4); screening through $\frac{1}{4}$ in. on 60 mesh, to (5) "ground rock," through 60 mesh, to (7).
4. Belt conveyor. To outer storage bins ready for market.
5. Belt conveyor. To screening bins to (6).
6. Rock-emery mill. Ground product to market.
7. Screw conveyor. To inside bins ready for market.

The total cost of mining and dressing "coarse crushed" is $1.975 per long ton. Cost for ground rock is $2.105 per ton.

Canadian Graphite Development[63].—A brief description is given of graphite milling at Port Elmsley, Ontario. The ore consists of graphite in limestone and gneiss. The ore is first roasted in an ordinary kiln to drive off the moisture; then it passes through a set of crushers and thence is treated by pneumatic jigs, where the flakes are separated from the gangue. After grinding in buhrstone mills the product is sized by screens into four grades for the market. The mill treats from one to two tons of ore per day. The material mined yields an average of 10 per cent. graphite.

Concentration of Lean Molybdenite[64].—Dr. J. Ohly describes a process which he experimented with for concentrating a molybdenite ore by means of surface tension of hard water. The ore contained 2.54 per cent. molybdenite and 97.25 per cent. quartz. The dry ore ground through 30 mesh was dropped gently by means of a bumping table upon an 18 per cent. solution of salts in water; the quartz gangue sank to the bottom, while the

[61] *Bulletin of Mines*, Canada.
[62] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 204.
[63] *The Canadian Mining Review*, Vol. XXV (1905), p. 122.
[64] *The Mining World*, Vol. XXIII (1905), p. 661.

molybdenite floated off and was passed over burlap and collected. A 97 per cent. extraction was obtained.

The Tungsten Industry of Boulder County, Colorado[65].—This article gives brief descriptions of the Nederland and Boulder county mills which treat the tungsten ores of Boulder county, Colorado. The ore is wolframite in gneissic rock. The scheme of treatment at the Boulder county mill is as follows: The pulp stamped through a 20-mesh wire screen goes to Wilfley tables, the entire product from which is pumped to Brown classifiers, where a three-grade classification is made before distribution to three Frue vanners. The slimes from the vanners go to blanket tables, obtaining an added saving. The concentration is about 12 into 1.

In the Nederland mill, using Wilfley slimers in place of vanners, a recovery of from 60 to 75 per cent. tungstic acid is made.

New Method in the Metallurgical Treatment of Copper Ores[66].—N. S. Keith describes a method for treating low-grade copper ores. The ore suitable for this process is one containing copper carbonates, sulphides, metallic copper and silver and gold values, with either quartz or iron oxide or limestone as gangue. The ore is crushed to 60 mesh more or less, according to the size of the mineral particles, and mixed with 3 per cent. finely ground coal, and then heated in a reverberatory furnace. The copper, silver and gold melt into metallic globules, and are afterwards extracted by running the gold over concentrating machines. The gangue is non-fluxing and does not fuse.

Magnetic Concentration in British Columbia[67].—A description is given of the zinc plant built by the Kootenay Ore Company for the concentration of zinc ores of Kaslo, B. C. The scheme of this custom mill is as follows:

1. Ore bins. To automatic feeders to (2).
2. White-Howell revolving furnaces. To (3).
3. Rotary cooling cylinders (air currents). To (4).
4. Preliminary sizing and recrushing of oversize products. To (5).
5. Elevated to top buildings. To (6).
6. Close classification. Product, eight sizes, to (7).
7. Eight separate bins. By automatic feeders to (8).
8. Magnetic separators. Finished zinc product to shipping bins; waste magnetic iron.

The White-Howell furnace has a capacity of 60 tons and upwards per 24 hours. The finished product contains about 50 per cent. zinc.

Magnetic Separation of Tin Ore.[68]—A description is given of a method employed at the Loudon mill, Irvinebank, to separate iron from tin

[65] *Mining Reporter*, Vol. LI (1905), p. 4.
[66] *Journal* of Franklin Institute, Vol. CLX (1905), p. 147.
[67] *Mining and Scientific Press*, Vol. XC (1905), p. 373. Abstracted from *Kaslo Daily News*.
[68] *Mining Reporter*, Vol. LII (1905), p. 491; *Mining and Scientific Press*, Vol. XCI (1905), p. 341, abstracted from North Queensland *Register*.

concentrates. The iron and tin concentrates, which are nearly of the same specific gravity, are fed over a vanner, the wash-water of which is charged with electricity; the iron particles becoming magnetic join together and offering resistance to water are swept off into the tailings; the tin is carried over the head of the vanner.

The Buchanan Magnetic Separator.[69]—A description of the Buchanan magnetic separator manufactured by the George V. Cresson Company, Philadelphia. The separator consists of a hopper with a vibrating chute which feeds the ore on a revolving cylindrical brass drum. A strong uniform magnetic field is produced for one-half the circumference by winding the wire around a form within the drum. The machine is made in sizes from 18 in. in diameter to 36 in., with capacities from 50 to 100 tons.

Imperial Magnetic Ore Separator.[70]—A description of a magnetic separator manufactured by the Imperial Ore Separator Company, Brooklyn., N. Y., is given. The separator consists of two rolls facing each other and revolving towards each other, with stationary magnets within each roll. The ore is fed between the rolls from above; the magnetic portion is held to the rolls and revolved to hoppers at the sides; the non-magnetic portion drops straight down into a hopper.

Magnetic Separation.[71]—A description of the Wetherill magnetic separator consists of two horseshoe magnets placed side by side and parallel to each other. They have cores of soft iron 8.6 in. long, 10.75 in. wide and 2.5 in. thick, wound with 950 coils, each wire carrying 6 to 8 amperes with 52 volts per magnet. The pole pieces are 15.2 in. long, 10.75 in. wide and 2.5 in. thick. The north and south poles are together at each end. Three belts are employed with each pair of magnets. The first belt brings the ore in horizontally under one pole piece of the upper magnet. The second belt operates under the surface of the upper pole piece, rounds the tips, and moves up the bevel of its wedge, bringing with it magnetic particles which, as they get above the magnetic field, fall off. The third belt accomplishes the same for the other pole piece of the upper magnet.

The Wetherill separator is used at the New Jersey Zinc Company's works for separating franklinite from willemite. The DeBeers Consolidated Mines, Kimberley, South Africa, use this separator to separate diamonds from the associated ferrous minerals, such as magnetite, chromic and titanic iron, etc.

The International Magnetic Separator.[72]—Frederick T. Snyder describes the magnetic separator made by the International Separator Company, Chicago. This separator consists of a cylindrical armature (made up

[69]*The Iron Age*, Vol. LXXV (1905), p. 1974.
[70]*The Mining World*, Vol. XXIII (1905), p. 99.
[71]*Ibid.*, Vol. XXIII (1905), p. 177.
[72]*The Engineering and Mining Journal*, Vol. LXXX (1905), p. 396.

of thin laminated disks of a special annealed wrought iron) mounted upon a steel shaft. Each disk is toothed like a circular saw. These teeth are so staggered on adjacent disks that the surface of the armature presents a multitude of points. This armature is mounted so as to revolve horizontally between the pole pieces of an inverted horseshoe magnet. The poles of this magnet induce magnetic poles on the teeth opposite them. Ore is fed on top of the revolving armature, non-magnetic particles are thrown off by centrifugal force, while magnetic particles are held by the concentrated power of the magnetized teeth. The induced magnetism of the armature gradually decreases as a certain element moves away from the pole of the horseshoe magnet; the armature gradually drops particles into a series of long narrow hoppers according to their attractability, until all drop off at the junction of the north and south field. Means of control are: varying the speed, thus varying the centrifugal force; varying the current, thus varying the magnetic force.

The author gives the following results obtained with this separator:

SEPARATOR RESULTS.

Description.	Product.	Weight Per Cent.	Zinc Per Cent.	Sulphur Per Cent.	Per Cent Saved.
Zinc Ore No. 1.	Orig.	100.	28.2	....	....
	Zinc	49.3	47.3	....	82.2
	Iron.	48.2	8.8	....	....
Zinc Ore No. 2.	Orig.	100.	36.3	....	....
	Zinc	70.	48.2	....	93.2
	Tailing.	28.	6.8	....	....
Zinc Ore No. 3.	Orig.	100.	38.4	....	....
	Zinc	66.5	49.6	....	86.2
	Iron.	32.4	9.4	....	....
			Manganese Per Cent.		
Manganese Ore No. 4.	Orig.	100.	14.6	....	....
	Manganese.	31.8	39.0	....	81.2
Manganese Ore No. 5.	Orig.	100.	15.8	....	....
	Manganese	13.2	48.2	....	83.4
	Tailing.	76.4	1.6	....	....
			Iron Per Cent.		
Hematite Ore No. 6.	Orig.	100.	46.		
	Iron	56.3	66.7	....	81.5
	Tailing.	43.7	19.4	....	....
Banded Jasper No. 7.	Orig.	100.	41.	....	....
	Iron	64.8	52.7	....	83.4
	Tailing.	35.2	12.9	....	....
Magnetite Ore (Crude.) No. 8.	Orig.	100.	55.2	1.94	....
	Iron	71.6	64.1	.28	83.1
	Sulphur.	28.7	25.87	7.2	....
Magnetite Ore (Washed).	Orig.	100.2	59.74	2.58	....
	Iron	96.2	56.39	.50	83.6
	Sulphur.	2.5	39.3	15.5	....

At present eighteen of these separators are in use at the Yak mill, Leadville, making a separation of unroasted pyrite, blende and pyrrhotite.

The Ball and Norton Belt Type Magnetic Separator.[73]—In a separator made by Witherbee, Sherman and Company, Mineville, New York, the Ball-Norton principle of a series of magnets alternating the poles, in connection with horizontal belts, is used. The separator consists of a feed-hopper with a feed-roll, allowing the ore to drop upon a horizontal feed-belt. This belt carries the ore up to the horizontal take-off belt, within which are a series of twelve magnets with opposite poles adjacent. These magnets hold the magnetic particles up against the take-off belt, constantly jigging them end for end, allowing the gangue to drop off into a tailing conveyor, until the belt carries them beyond the last magnet, where they drop off into the concentrates conveyor. This machine has been used on phosphorus magnetic ore containing from 1.35 to 2.25 per cent. phosphorus, giving a finished product containing 0.6 per cent. phosphorus and 66 to 67 per cent. iron. The capacity is from 20 to 25 tons per hour on material through a ¼ in. sieve and under.

Electro Magnetic Separation.[74]—Results are given of the Mechernicher separator on a blende-pyrite ore at the zinc works at Lipine, Upper Silesia. The ore contained from 25 to 26 per cent. zinc and a large amount of iron. One and one-half tons of the roasted ore gave 0.9 ton blende (40 per cent. zinc) and 0.6 tons of iron product (15 per cent. zinc).

Elmore Process for Diamond Recovery.[75]—The editor states that successful experiments were made with the Elmore oil process on diamonds at the Premier diamond mines, practically 100 per cent. of the diamonds being recovered. As a result, the Premier diamond mines are going to erect immediately an oil concentration plant capable of treating 800 tons per day. It is thought this process will give a higher extraction, lower cost of treatment, and protection from theft.

Oil Concentration Process in British Columbia.[76]—In the annual report of the Le Roi No. 2 mine statement is made that the treatment of tails from Wilfley, running $3.23 in gold, silver and copper, by the Elmore oil process was a commercial failure. Every dollar's worth saved cost two dollars to effect the saving.

The Elmore Process in Rossland.[77]—Mr. Elmore, in a letter, states that the exact reasons for the failure of the Elmore process in Rossland were the low grade of ore treated and insufficient supply of ore to keep mill running to full capacity.

Sutton-Steele Electrostatic Magnetic Separator.[78]—The machine consists of an ore hopper, a belt with a metallic coating, one or more magnetic rolls and the same number of auxiliary cleaning rolls. The ore is fed

[73]*The Iron Age*, Vol. LXXVI (1905), p. 1367.
[74]*Zeitschrift für Berg- Hütten- und Salinenwesen*, Vol. LIII (1905), p. 125.
[75]*Ibid.*, Vol. LXXX (1905), p. 257
[76]*Mining Reporter*, Vol. LI (1905), p. 144.
[77]*Canadian Mining Review*, Vol. XXIV (1905), p. 177.
[78]*Engineering and Mining Journal*, Vol. LXXX (1905), p. 253.

on the traveling belt, whose metallic coating has been given a static charge. The ore particles oscillate up and down between the belt and the magnetic rolls; the magnetic particles are attracted to the rolls and are removed by the cleaning rolls; the gangue falls back on the belt and is discharged.

Concentration of Iron Ores.[79]—Dr. Weiskopf gives illustrated descriptions of the following magnetic separators: Monarch (Ball-Norton), Grondal (three types); Heberle (wet); Frodings, Erikson, and Forsgrend. He also gives a table of results obtained in mills using magnetic concentration of iron ores:

Place and Description of Ore.	Type of Separator.	Size of Largest Particle. m. m.	Product.	Orig. Weight Per Cent.	Iron Per Cent.	Phosphorus Per Cent.	Sulphur. Per Cent.	Capacity Per Day. Tons.	Capacity Per Week. Tons.	Horse Power. H. P.	Amperes.	Volts.
Herrang, Pyroxene and Granite with Magnetite.	Monarch.	8	Orig. Concs. Tailing	100. 55. 45.	45. 60. 15.	.003 .003	2.0 0.5	10.	800	260	8–10	100
Svartonbei Lulea. Gellivara Ore.	Monarch.	2	Orig. Concs. Tailings	100 85.1 14.9	58. 70. 25.5	1.0 0.127 13.7*	....	......	2000 to 2500	230	7	100
Bagga, Hematite, Quartz and Amphibole.	Grondal.	....	Orig. Concs. Tailings	100. 63.7 36.3	35. 61.	....	....	......	....	....	18–10	35
Strassa Dump Ore.	Grondal.	.... 1	Orig. Concs. Tailings	100. 45.5 54.5	36.8 61.58 12.	0.014 0.006	0.11 0.045	30 to 40	....	30 to 35	1.7	30
Klacka. Poor Iron Ore with Hematite.	Grondal.	77 % is 0.15	Orig. Concs. Tailings	100. 45.9 54.1	38 58. 13.6	....	....	20	....	20	8.5	.....
Persberg Dump Ore.	Grondal.	.... 5	Orig. Concs. Tailings	100. 21. 79.	15to20 57.	....	....	25 to 30	....	55	5–7	30
Bredsjo Dump Ore.	Grondal.	.... 1.5	Orig. Concs. Tailings	100. 48.6 50.	45.3 64. 7.	0.0083 0.0023 0.0024	0.198 0.082 0.296	30.8	....	40	....	.....

*This is $P^2 O^5$.

The author gives the following system as a combination of magnetic and wet treatment used in several mills:

1. Raw ore to (2).
2. Gates breaker. To (3).
3. Ball-mill (screen 1 mm.). Over 1 mm. to (3); under 1 mm. to (4).
4. Grondal separators. Magnetic to smelter; non-magnetic to (5).
5. Jigs. Concentrates hematite to smelter; tailings to waste.

Oil Concentration.[80]—A. Schwartz obtained a patent (No. 23,906, 1904) which consists of adding fatty substances to petroleum and also sulphur compounds, in order to increase the efficiency of the process for concentrating by means of oils.

Electrostatic Concentration.[81]—Lucien T. Blake discusses electrostatic concentration. He states that the specific time element of a particle

[79] *Stahl und Eisen*, Vol. XXV (1905), pp. 471, 532.
[80] *Engineering and Mining Journal*, Vol. LXXIX (1905), p. 590.
[81] *Ibid.*, Vol. LXXIX (1905), p. 1036; *Trans.* of the American Electro-chemical Society.

for acquiring a charge and a given potential depends upon the conductivity of the particle and its capacity factor. The author gives the following table of results obtained with electrostatic concentration:

Description.	Product	% Zinc	% Lead	% Iron	Remarks
Idaho, lead and zinc table concentrate.......	Orig. Lead Zinc	25.87 4.45 45.11	20.40 50.58 3.81	12.02 16.14 7.40	High Silver.
Nevada, chalcopyrite in garnet.............	Orig. Conc. Tailings	*Copper* 3.20 12.94 0.40	....	....	12 mesh. Saving 81% copper. Crude ore, no dust removed.
Arizona, copper carbonate in limestone......	Orig.	4.60 33.89 0.79	....	....	20 mesh. Saving 70% copper. Crude ore, heated to redness.
Michigan, native copper in sandstone.......	Orig. Conc.	2.68 38.70 0.27	....	....	30 mesh. Saving 76% copper.
Utah, chalcopyrite in heavyspar......	Orig. Conc. Tailings	6.06 26.81 0.93	....	....	Saving 88% copper.
New Mexico, gold-bearing pyrite in quartz ...	Orig. Conc. Tailings.	1.14 4.20 0.33	11.60 36.12 1.27	0.80 2.42 0.14	Saving 74% gold.

The author also states that in separating molybdenite from gangue a concentrate containing 96 per cent. molybdenite was obtained. The author gives a description of the Blake-Morscher machine. He also gives a list of the minerals which are conductors and non-conductors, as obtained from practical experience by W. G. Swart.

Blake-Morscher Ore Separator.[82]—The separator is constructed almost entirely of pine timber, and is 10 ft. 6 in. long, 8 ft. high and 3 ft. 4 in. wide; as a rule two are mounted together so as to form one unit. The ore is fed by means of a small cup elevator at one end of the separator into a box, which is provided with a screw conveyor, acting as a distributor for the two feed-hoppers, which are slotted (¼ in.) for their entire length. From here the ore drops on to feed-boards which are adjustable to various degrees of inclination by means of eye-bolts and turn-buckles. Adjustable gates are used over the ends of the feed-boards. The ore drops from the feed-board upon the revolving repelling poles. The separator has four poles arranged in pairs above each other; the two upper poles are negative and the two lower poles are positive. Under the poles is a system of pockets collecting three products; viz., good conductors, lesser conductors (middlings) and non-conductors. The poles are adjustable as to speed, and are scraped on the back side by a scraper-board. The middling product is elevated by the cup-elevator back into the bin, to be fed over again. The author gives a description of the Wagner mica plate used to furnish the static electricity,

[82] *Trans.* of Institution of Mining and Metallurgy, Vol. XIV (1904–5), p. 169; *Engineering and Mining Journal*, LXXIX (1905), p. 1036; *Mining Magazine*, Vol. XI (1905), p. 515.

and also states that 3 h.p. is required to drive one static machine and two separators, including the elevators. The author gives the following practical working results:

Description	Product	%	% Zinc	% Lead	% Iron	% SiO_2	Oz. gold per ton.	Oz. silver per ton.
Gilpin Co., Colo. Bonanza mine........	Orig.	100.	29.0	5.5	22.8	5.2	0.24	6.56
	Iron	36.	6.2	10.0	40.0	2.0	0.37	8.13
	Zinc	63.	44.0	2.2	10.9	6.8	0.16	5.24
Leadville ore[1]. Wilfley table middlings.	Orig.	100.00	30.40	4.60	21.00	3.38	0.04	6.80
	Iron	47.55	7.37	9.40	36.23	3.90	0.06	8.10
	Zinc	51.80	51.10	0.60	7.97	2.85	trace	5.10
Colorado Zinc Co., Denver. Middlings from Wetherill separator..........	Orig.	100.00	15. to 18.	4.	35. to 37.	3.	trace	10.
	Iron[2]	75.	7.2	5.5	40.	1.3	trace	12.
	Zinc[3]	25.	47.2	0.8	....	3.3	trace	4.
Joplin, Mo. Jig concentrates. 20 mesh...	Orig.	100.	....	....	....	....	....	....
	Iron	....	4.	....	....	....	....	....
	Zinc	....	..60.	....	....	....	....	....
Wisconsin. Jig concentrates. 10 mesh.	Orig.	100.	22.	....	....	....	....	....
	Iron	63.	3.	....	....	....	....	....
	Zinc	36.	59.	trace	4.	....	....	....
Stolberg, Germany.. Middlings. 30 mesh.	Orig.	100.	36.9	....	....	....	....	....
	Iron	39.	8.2	....	....	....	....	....
	Zinc[5]	61.	54.3	....	....	....	....	....

[1]$8 per ton net profit after paying for the material and all expenses connected with the separation. [2]Sold to the smelter for $7 to $8 per ton. [3]Sold at Denver for $15 per ton. [4]Net profit of $9 per ton from an otherwise worthless product. Expenses 50 cts. per ton. [5]Saving of 90.2 of zinc.

The Silver Ledge Mill.[83]—W. G. Swart describes the method of concentration at the Silver Ledge mill, 8 miles north of Silverton, Colorado. The ore consists of galena, blende and pyrite, with a sprinkling of chalcopyrite disseminated through altered rhyolite. The ore also contains gold and silver values. The following system is used at the mill:

1. Ore from mine is dumped into large storage bins. To (2).
2. Blake breaker (10x20 in.). Product by shaking launder to (3).
3. Davis rolls (16x36 in.). Product (½ in.) to storage bins to (4) by means of plunger feeders.
4. Two 6-ft. Chili mills (24 mesh or 1 mm.). Product sand pumped to (5).
5. Hydraulic classifier. Five products to (6).
6. Five Wilfley tables. Lead concentrates to (7); middlings to (8); tailings to (9).
7. Jigger room. To smelter.
8. "Pony" Wilfleys (one for each large Wilfley). Lead concentrates to (7); tailings to (9).
9. Five Wilfleys. Lead concentrates to (7); middlings to (10); tailings to waste.
10. Five "pony" Wilfleys. Lead concentrates to (7); zinc middlings to (11); tailings to waste.
11. Shaking conveyor. To rotary dryer to (12).

Mining Reporter, Vol. LI (1905), p. 68.

12. Blake-Morscher separator. Zinc concentrates to smelter; lead concentrates to smelter.

COAL WASHING.

The Spring Valley Coal Company's Mines.[84]—Leo Gluck describes the mining and briefly the washing of coal at these mines. Each of the mines has three shaker screens and two gravity screen plants to separate the coal into the following sizes: Chunks 6 in. and over; lump, $1\frac{1}{4}$ in. and over; egg, 6 in. to $1\frac{1}{4}$ in.; nut, $\frac{1}{4}$ in to $\frac{7}{8}$ in.; slack, $\frac{7}{8}$ in.

The mine fine coal goes to the washery, where the following system is in use:

1. Fine coal from cars elevated to (2).
2. Revolving screens. Oversize to (3); under to (5).
3. Bin. To (4).
4. Stewart jigs. Products to revolving screens drained of excess of water, to shaking screens, washed with clear water, to bins for shipment.
5. Bins to Stewart jigs. Products treated same as (4).

The capacity of this washery is 100 tons per day.

The Separating of Dust from Coal.[85]—An illustrated description is given of an apparatus used in separating the dust from coal. The coal is fed from a hopper down a vertical chute and drawn away by a conveyor at the bottom. While passing down the chute, the coal is subjected to suction of a fan on one side, which draws the dust-laden air through the coal, passes it through a trapping or catching chamber, and forces it around and through the coal from the other side. This apparatus is in use at two collieries in Scotland and one in Staffordshire.

Washery of the Bethune Mining Company.[86]—A description is given of the washery of the Bethune Mining Company in the department of Pas de Calais, France. At the washery the coal is elevated to a table sieve with 50 mm. openings. The oversize goes to the hand-picking department; undersize goes successively to screens with openings 30, 15, 8 and 3 mm. Product 3 to 5 mm. goes to fine jigs, the tailings from which go to two other jigs to be washed again. Products 8 to 15, 15 to 30, and 30 to 50 go to the coarse jigs, where coal is separated from the waste. The product below 3 mm. is put through a system of spitzkasten and jigs to clean the coal. The washery has a capacity of 130 tons per hour.

Coal Washer.[87]—An illustrated description is given of a machine used for washing small coal at the Friedrichsthal mine, Saarrevier, Prussia. The washer is essentially a revolving water-washed trommel fed by a screw conveyor, which is in turn fed by a bucket elevator. The capacity is 11 tons per day.

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[84] *The Mining World*, Vol. XXII (1905), p. 435.
[85] *The Colliery Guardian*, Vol. LXXXIX (1905), p. 608.
[86] *Gluckauf*, Vol. XLI (1905), p. 842.
[87] *Zeitschrift für Burg-, Hütten- und Salinenwesen*, Vol. LIII (1905), p. 127.

Rotary Screen on Fixed Shaft.[88]—This screen, used at new plant of the Treverton Coal Land Company, Pennsylvania, differs from an ordinary revolving screen in that the shaft is fixed, and the water is fed internally from two 3-in. pipes running near and parallel to the shaft. A saving in the amount of water used and greater efficiency is claimed.

Truesdale Breaker and Washery[89].—This article gives a description of the new double coal-breaker and washery of the Delaware, Lackawanna and Western Railroad. Longitudinal and transverse sections of the breaker are given. The breaker separates into steamboat, grate, egg, stove and chestnut sizes. When completed it will handle 4000 tons per day. The coal starts at the top of the building, where it passes over stationary bars, then to oscillating bars and picking bands in the chipping-room. It then passes through six sets of screens, through mechanical pickers and by conveyors into storage bins for the different sizes. The screens are cylindrical double jacketed, making only two sizes and are in duplicate (three different sizes), 24 in all. The pickers are of the Emery type for first picking, and are followed by Nichter spiral picker. The coal passing through the chestnut screen goes to the washery. The machines of the breaker are equipped with independent motors of the alternating current induction type.

Shaking Screens of the Truesdale Washery[90].—In this article is given a description of the shaking screens used at the washery of the Truesdale plant of the Lackawanna Company. All coal in the breaker passing through chestnut screen and also all "bony" from the pickers (this last being rebroken) goes to the washery. Here the sizing is done by two shaking screens. The chestnut and pea after leaving the shaker go to spiral pickers for the removal of slate, and then to pockets. The buckwheat sizes go direct from screens to the pockets. The screens are arranged five in a group, superimposed one on the other; all are driven from same shaft, by pairs of eccentrics set 72 deg. apart to neutralize the vibration. To further balance the screens they are supported at the eccentric end by triangular hangers; the apex of each hanger is pivoted on the screen, while each corner of the base is hung from adjustable springs. The two top screens are 5 ft. 9 in. wide, the three bottom ones are 6 ft. 2 in. The top screen is 30 ft. long; the rest decrease in length uniformly to the bottom one, which is 25 ft. long. The screens have a pitch of 1 in 12. A 19 h.p. motor gives the eccentric shaft 150 r. p.m.

The Nichter Spiral Slate Picker at the Truesdale Breaker[91].—This consists of a spiral chute wound around a vertical shaft, with another larger spiral chute winding around the smaller one. The coal and slate

[88] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 347.
[89] *Ibid.*, Vol. LXXX (1905), p. 584.
[90] *Ibid.*, Vol. LXXX (1905), p. 867.
[91] *Ibid.*, Vol. LXXX (1905), p. 735.

are fed at the top into the smaller spiral and run down the chute by the force of gravity; at the same time the chute is being moved away from coal and slate by revolving around the shaft. The slate clings near the shaft, while the lighter coal tends to work toward the outer edge and drop over into the larger spiral chute. The spiral is driven by a leather wheel bearing on a friction disk attached to the shaft. By varying the distance of the leather wheel from the shaft the speed is varied to suit the different materials.

Breaker Screens[92].—This article discusses the relative advantages and disadvantages of shaking and cylindrical screens. The rotary type has the advantage that it runs smoothly and clears itself readily of coal wedged in the mesh, also that it is best for preparing coal "dry." It has the disadvantages that it occupies much space, breaks the coal, and uses only one-quarter of its screening surface at any instant. The advantages of the shaking type are minimum of breakage and great efficiency in preparing small sizes (pea and buckwheat). The disadvantages are vibration, racking the building (which can be reduced to a minimum by opposing the screens in pairs), and lodging of coal in meshes which has to be picked out by hand.

Coal Washing[93].—R. A. Henry discusses the theoretical and experimental side of the separation of coal from slate, showing mathematically and diagrammatically the susceptibility of different mixtures of coal and slate to separation by washing. He also describes with illustrations the different types of jigs used in washeries.

Jigs at the Pittsburgh Coal Washer[94].—This is a description with the plan and elevation of the coal washery used by the Pittsburgh Coal Washer Company to wash coal free from sulphur and ash (slate). The primary washing system consists of one or more drop-motion reciprocating valve-controlled bottom jigs. These jigs run at 35 strokes per minute (twice as fast down as up), with the length of stroke adjustable. The jig-box has two bottoms, one perforated, the other 12 in. below, solid, with 12 cast-iron flap valves and seats. On the downward motion of the jigs the valves open and allow the water to rush in and raise the coal, slate and sulphur; on the upward motion of the jig they close and hold the water in the jig, allowing the material to stratify in comparatively still water. In front of the jigs are gates arranged to draw off the slate and sulphur. In the bottom of the jig tank is a screw-conveyor to carry the fine slate, sulphur and any coal which passed through the screen and to deliver it into the sludge tank. The coal flows over the front of the jig into a screen, from which it is taken by a conveyor and elevated to the clean coal-bins. The secondary washer is a duplicate

[92] *Engineering and Mining Journal*, Vol. LXXX (1905), p. 592.
[93] *Revue des Mines*, Series IV, Vol. X (1905), p. 274.
[94] *Iron Age*, Vol. LXXVI (1905), p. 1234.

of the primary as far as jigs are concerned, but instead of screens the washed coal goes into settling-tanks. With this apparatus one man and one boy can wash 100 tons in eight hours.

Bituminous Coal Washing[95].—L. A. Harding and G. R. Delamater discuss the standard methods of preliminary investigation of coals to be used in coking, also the different machines used in the preparation of coal. One of the methods of preliminary investigation is to put a crushed sample of the raw coal into a solution of calcium chloride with a specific gravity of 1.35. The shale and pyrite sink, while the coal floats. The two products are analyzed as to their sulphur, ash, volatile matter and fixed carbon contents. From these analyses, and the known limit of sulphur and ash allowable in the coke, the limit of washing can be determined.

Various styles of crushers and rolls are described. The Cornish tooth-rolls used to break down the run-of-mine coal preliminary to a fine crushing gives the following results:

Size of Rolls.	Maximum Size of Lumps.	Final Product.	Capacity, Tons per Hour.	H.P. Required.
	Inches.	Inches.		
17 in. diam. by 24 in. long..	10x10	$2\frac{1}{2}$ and less	30	5
28 in. diam. by 24 in. long.. (Coarse long tooth)	12x20	4 and under	50	6 to 10
28 in. diam. by 24 in. long... (Small tooth)	12x20	$2\frac{1}{2}$ and under	40	6 to 10
28 in. diam. by 36 in. long...	Run of mine	$2\frac{1}{2}$ and under	70	10 to 15
29 in. diam. by 32 in. long...	Run of mine	4 and under	90	20

The two-roll disintegrator is used to break the coal from 3 to 4 in. or more down to $\frac{3}{4}$ in. These rolls generally run at differential speeds in a ratio of $1\frac{1}{2}$ to 1. This introduces a tearing action in addition to the crushing action, and gives better results. The hammer crusher consists of a series of hammers pivoted to a shaft revolving at 600 to 1200 r.p.m. and surrounded by bars set with a clear opening between them equal to the maximum size of product desired. These machines are not desirable for coal-washing plants, as they use more power than a set of rolls with a corresponding capacity and produce a large percentage of very fine coal, which is undesirable. There are two types of feeders used, the revolving pocket cylinder type and the reciprocating type; of these the latter is the best, giving a uniform feed. Two methods are used in classifying coal, screens and hydraulic. Screens give perfect sizing, but have the disadvantage of short life. In a simple classifier the coal and water coming in at one end are forced into the first pocket by a diving-board, where an upward current of water carries over into the next pocket all but the large pieces, which sink to the bottom and

[95] *Mines and Minerals*, Vol. XXV (1904–5), pp. 451, 538, 577.

are drawn off to the jigs. There are three methods of drying washed coal—sludge recovery system, dry screens and drying bins; of these the last is preferable, because there is no loss of fine coal and the drained water may be used over again.

The Preparation of Anthracite[96].—The new Hammond breaker built by the Philadelphia and Reading Coal and Iron Company is 376 ft. long, 155 ft. wide and 128 ft. high, and has a capacity of 2000 tons of coal per day. The cars of coal are delivered from the mine in a steady stream to the tip level by means of a chain hoist. The coal is tipped over bars, spaced 5 in., sloping 5 in. in 1 ft. The large material passing over the bars goes to the picking platform, where rock is put in one chute, the pure coal in another, and the mixed coal and slate in still another chute. The coal and mixed coal and slate go to separate toothed rolls ($30\frac{1}{2}$x52 in.), the products of which go to shaking screens ($4\frac{1}{2}$x$3\frac{1}{2}$ in.). Material over $4\frac{1}{2}$ in. is "steamboat" and goes to picking platform. Material through $4\frac{1}{2}$ in. and on $3\frac{1}{4}$ in. is broken coal and goes over picking tables to rolls (26x52 in.), and is broken into egg size; the product goes over shaking screens ($2\frac{1}{4}$ in.) and directly to pockets. Material through $3\frac{1}{4}$ in. goes successively to shaker screens with openings $2\frac{1}{4}$, $1\frac{1}{2}$, $\frac{7}{8}$, $\frac{9}{16}$, $\frac{5}{16}$, $\frac{3}{16}$, and $\frac{1}{8}$ in.

The products of these screens go to separate jigs. From the jigs the coal passes over slate shakers, which take out the flat slate and coal and the small coal produced by breakage in the jigs. The chestnut (over $\frac{7}{8}$ in.) and smaller sizes go directly from the slate shaker to the pockets, while the grate ($1\frac{1}{2}$ in. and egg $2\frac{1}{4}$ in.) pass first over picking-tables. The flat coal and bony coal picked out at various places is put through rolls and passes again through the series of screens.

The rolls used in the breaker have square pyramidal teeth, and are made of nine manganese steel segments to the round. The shaker screens are double, the upper screen is 4 ft. by 11 ft. 6 in., and the lower screen is 5 ft. 9 in. by 8 ft. 8 in. The pitch is $\frac{7}{8}$ in. per foot, the throw 6 in. and the revolutions of driving shaft 140 per minute. The jigs are double two-compartment jigs. with two plungers and four sets of mesh-plate screens. The stroke of the piston is 6 in. for broken coal and 5 in. for all other sizes. The revolutions of the driving shaft are 100 per minute. The capacity varies according to the size of coal jigged from 120 to 200 tons per 9 hours. The breaker uses 1070 gallons of water per ton of output.

The Old Hammond Breaker[97] is also described, to show the changes that have taken place in coal-breakers.

A Bituminous Coal-Breaker[98].—Lewis Stockett describes a coal-breaker recently erected at Stockett, Montana. The coal coming from the mines in cars, 1.5 tons capacity, is weighed automatically and dumped over a

[96] *Mines and Minerals*, Vol. XXV (1904-5), p. 280.
[97] *Ibid.*, Vol. XXV (1904-5), p. 382.
[98] *Trans.* of the American Institute of Mining Engineers, Vol. XXXV (1905) p. 31.

grizzly 12x6 ft. sloping 1 in 2 and having 2-in. spaces. The undersize goes to shaking screens, having 1-in. round perforations, 6-in. throw and 100 strokes per minute, which remove the slack from the coal. The slack goes directly to a railroad car. The oversize from the shaking screen is elevated to the top of the building. The oversize from the grizzly falls on a traveling belt, where the impurities are picked out by men; the coal drops into rolls crushing to 4 in. size. These rolls are of removable tooth type, 36 in. diam. by 48 in. wide, and make 75 r.p.m. The crushed coal is elevated to shaking screens 5 ft. by 46 ft. sloping 1 in 4 and making 100 6-in. strokes per minute. The plates of the screens have respectively 1, 1.5, 2, 2.5, and 3 in. round perforations and separate the coal into slack, pea, nut, stove, egg and broken sizes. The slack is clean and goes direct to the mixed coal-bin. The other sizes pass to spiral pickers, from which the coal goes to picking belts, where it is further freed from impurities. It then goes to bins ready for the market. The breaker received 2000 tons from the mine daily, 300 tons of which is slack and 200 tons impurities. The total additional cost of putting the coal through the breaker is $0.06 per ton.

Labor Saving Devices in Coal Mining[99].—R. V. Norris discusses coal-breakers, shaking screens, jigs, and spiral and chute pickers. He states that in the last few years the practice has been to construct the rolls used in breaking coal after the segmental type, with cast manganese steel segments, either longitudinal or circumferential. This type is much to be preferred over the insert tooth type, especially for small size coal, where a pair of bone rolls contains 2100 teeth. The author also states that the tendency has been in all modern plants to replace round screens by shaking flat screens, balanced in pairs. The advantages claimed are lower first cost, lower repairing charges and increased capacity. The author briefly describes the two types of jigs—the plunger jig and the pan jig; he calls these wet pickers. The author mentions various dry pickers, such as the Fern, Emery and Pardee spiral. The Fern picker, which is suitable for flat slate, is a simple angle bar with tapered slot, and with a drop opening at the end. It has a capacity of 4 tons per hour per foot of width. The Pardee spiral consists of sheet-iron spirals with adjustable pitch, so arranged that by centrifugal force the free-running coal is thrown over the edge of the inner spirals and caught in the wide outside casing; the bone runs near the outer edges of the small spirals and is delivered into a separate chute from the slate, which by reason of its poorer sliding qualities has a tendency to run close to the axis. The capacity is about 4 to 6 tons per hour, and the separation of coal and slate is good.

Breaker for Producing Small Coal.[100]—M. J. Julin describes briefly the

[99] *The Engineering Magazine*, Vol. XXVIII (1905), p. 553.
[100] *Annales des Mines de Belgique*, Vol. X (1905), p. 968.

breaker constructed by Humboldt at Kalk, near Cologne. The scheme of the mill is as follows:

1. Coal wagons. To (2).
2. Shaking feed table. To (3).
3. Screen (100 mm.). Oversize to (6); undersize to (4).
4. Screen (50 mm. opening). Oversize to (7), undersize to (5).
5. Screen (20 mm.). Oversize to bin; undersize to separate bin.
6. Crusher. To (8).
7. Crusher. To (8).
8. Screen (70 mm.). Oversize to bin; undersize to (9).
9. Screen (55 mm.). Oversize to bin; undersize to (10).
10. Screen (30 mm.). Oversize to bin; undersize to (11).
11. Screen (20 mm.). Oversize to bin; undersize to bin.

This plant is driven by an electric motor of 35 h.p. The author is uncertain as to its capacity.

IMPROVEMENTS IN SAMPLING AND ASSAYING.

By H. O. Hofman.

Sampling Ores.—C. S. Palmer[1] publishes some general remarks about the different degree to which components of an ore are liable to be reduced in size by crushing, which is one of the main difficulties in obtaining a correct sample.

A. T. French[2] experimented upon the "canvas," "alternate shovel," and "coning and quartering" methods of sampling down ore from 100 ℔. to 5-10 ℔. The ore was a copper-bearing pyrite with a silicious gangue; when broken, the finer parts were richer than the coarser. The original sample was crushed in a rock-breaker to a limiting size of 2 in. cube; then broken by hand to 1 in. cube and divided twice. The resulting 25-℔. sample was then broken to 0.5 in. cube and divided twice to furnish the final laboratory sample. The canvas method consists in mixing the crushed ore by rolling on a cloth about 6 ft. square, bringing it together in the center and shaking the four corners when gathered to free the cloth from dust, dividing the sample into four parts, etc. In the alternate shovel method the broken ore is shoveled over twice to a cone, and this reduced before the sample is taken. The coning and quartering method is the one that has been employed for many years. With two samples the following figures were obtained:

Canvas method.		Alternate shovel method.	
Original.	Duplicate.	Original.	Duplicate.
2.90 % Cu.	2.95 % Cu.	2.90 % Cu.	3.00 % Cu.
3.57 " "	3.57 " "	6.37 " "	7.45 " "
3.48 " "	3.48 " "	1.57 " "	1.70 " "
10.90 " "	10.70 " "	2.05 " "	1.90 " "

The above figures show the surprising result that the canvas method is more regular than the alternate shovel. With two other samples, the three methods were tried, giving the following results:

Canvas.		Alternate shovel.		Coning and Quartering.	
Original.	Duplicate.	Original.	Duplicate.	Original.	Duplicate.
8.97 % Cu.	9.07 % Cu.	9.45 % Cu.	8.67 % Cu.	8.87 % Cu.	8.77 % Cu.
9.83 " "	9.76 " "	9.96 " "	10.07 " "	9.02 " "	9.02 " "

The author is unfortunate with his mode of procedure with the alternate

[1]*Engineering and Mining Journal*, 1905, LXXX, 405.
[2]*Transactions* of Institution of Mining and Metallurgy, 1905, through *Mining and Scientific Press*, 1905, LXC, 271.

shovel, as in principle and in practice it is far to be preferred to coning and quartering.

H. C. Parmelee[1] gives some assay results of sampling gold ores which are interesting, but do not give sufficient information to draw conclusions about the methods of mixing and sampling, and the degree of fineness to which a sample ought to be ground.

W. D. Verschoyle[2] gives an illustrated description of a very simple sampling device, which resembles in principle the well-known Jones sampler.

H. Louis[3] describes a continuous sampling apparatus consisting of three V-shaped troughs. Trough 1 is inclined and receives the ore, delivers it to trough 2, also inclined but in an opposite direction; trough 2 discharges the ore upon the apex of an inverted trough, 3, placed horizontally, which divides the stream into two equal parts. It is not quite clear what advantage this apparatus for sampling offers.

A. Gullberg[4] patented an automatic ore-sampling machine with oscillating cutters, which resembles in principle the old form of Brunton sampler.

The Sturtevant Mill Company[5] has brought out a small size of its roll-jaw crusher for laboratory purposes, which is slightly changed to make cleaning easy. The crusher is made in two sizes, having jaw-openings 2x4 and 2x6 in., weighs about 400 lb. and has a capacity of 250 to 600 lb. ore per hour, crushing to sand-size.

O. J. Reynolds[6] describes the method of sampling a gold-silver-lead ore at the Helena works of the American Smelting and Refining Company, East Helena, Mont. The car with the ore having been weighed, the moisture-sample is taken by digging a hole into each end of the ore until about 15 lb. have been obtained. It is crushed to pass a $\frac{3}{8}$-in. ring, and 1000 grams are weighed out and dried at 100 deg. C. for about 24 hours. In the case under consideration, the moisture amounted to 1.8 per cent. In order to take the assay sample, the lot is unloaded, passed through a Blake crusher and through corrugated coarse rolls and a trommel with $\frac{3}{8}$-in. holes; any oversize is crushed in a pair of fine rolls and fed into the preceding trommel. The crushed ore is now put through an intermittent mechanical sampler (name not given) which cuts out $\frac{1}{5}$, or about 1500 lb., as a sample, to be further reduced to quartering. For this purpose it is dumped in a circle about 6 ft. in diameter and coned twice before the cone is flattened out to a thickness of 6-8 in. to be quartered. The ensuing sample is shoveled to form a ring, coned, quartered, and the operations repeated to the marking off of the quarters, which are denoted as X, XX, XXX, XXXX. Each quarter is then coned and quartered until it has

[1] *Western Chemist and Metallurgist*, 1905, I, 183.
[2] *Engineering and Mining Journal*, 1905, LXXX, 485.
[3] *Transactions* of Institution of Mining and Metallurgy, 1905, through *Engineering and Mining Journal*, 1905, LXXX, 61.
[4] U. S. Patent No. 782,235, Feb. 14, 1905.
[5] *Engineering and Mining Journal*, 1905, LXXIX, 347.
[6] *Mining Reporter*, 1905, LI, 371.

been reduced to 250 lb.; it is then sprinkled with water, if dusty, crushed in fine rolls to 14-mesh, reduced by quartering to about 20 lb., and finally passed through a sample-grinder to the limiting sieve of 60-mesh. The time required to sample the above lot was six hours: five men on the mechanical sampler for 1.75 hours; three men coning and quartering, three hours; two men on rolls and grinder, 1.25 hours. The following are the assays of the four samples:

Sample.	Au Oz. per ton.	Ag Oz. per ton.	Pb %	Zn %
X	7.12	41.1	34.5	12.1
XX	6.56	38.7	34.5	....
XXX	5.48	40.7	34.5	....
XXXX	5.76	39.0	34.5	....
Average.	6.155	39.87	34.5	12.1

The great irregularity in the values for gold is due to coarse scales which were ground with the pulp through a 100-mesh sieve and mixed in; also to the coarseness of the laboratory sample (60-mesh).

Sampling Base Bullion.—J. Asbeck[1] cast a bar weighing about 110 lb. from argentiferous lead of which a dip sample assayed 68.54 oz. per ton. He then cut the bar into three pieces and took samples as indicated by

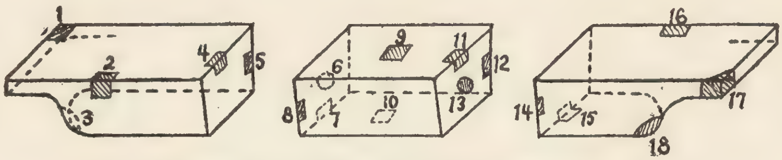


FIG. 1.—DISTRIBUTION OF SILVER IN A BAR OF ARGENTIFEROUS LEAD.

the numerals in Fig. 1. These gave the following results, in ounces per ton:[2]

No.	Ag Oz.	No.	Ag Oz.	No.	Ag Oz.	No.	Ag Oz.	No.	Ag Oz.	No.	Ag Oz.
1	78.46	4	71.16	7	77.87	10	72.91	13	65.87	16	69.85
2	75.25	5	71.29	8	78.16	11	62.70	14	70.58	17	70.43
3	72.18	6	67.52	9	69.41	12	68.65	15	65.04	18	69.41

They show that the distribution of silver is uneven, those parts are poorest which solidify last.

J. W. Root[3] discusses the sampling of base bullion by punching, sawing, dipping from the kettle and casting from the molding siphon. Sections of bars cooled slowly and quickly give information as to the distribution of silver.[4]

[1] *Chemiker Zeitung*, 1905, XXIX, 78.
[2] For other data on this subject see Hofman, "Metallurgy of Lead," 1899, 348.
[3] *Western Chemist and Metallurgist*, 1905, I, 221.
[4] For similar data, see Raht, THE MINERAL INDUSTRY, 1894, III, 414.

Assaying: New Publications.—P. H. Argall published a book, "Western Mill and Smelter Methods," which is a reprint with amplifications of the author's paper, "Smelter and Mill Methods of Analysis in Use in the West," previously referred to.[1]

A. H. Low brought out a book, "Technical Methods of Ore Analysis," J. Wiley & Sons, New York, $3, in which the metallurgical chemist will find just what he usually seeks and looks for in vain, viz.: the description of a reliable method for determining a metal or compound with so much detail that, with the usual care, he can obtain a result that is satisfactory to everybody concerned.

Assay Laboratory.—J. L. Buskett[2] describes with plan, side and end elevations an assay office for a mine in which mainly gold and silver assays

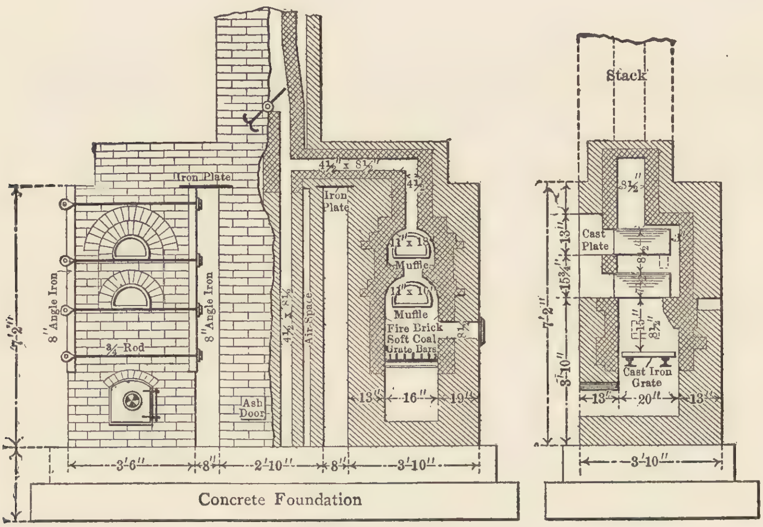


FIG. 2.—TWO-MUFFLE ASSAY FURNACE.

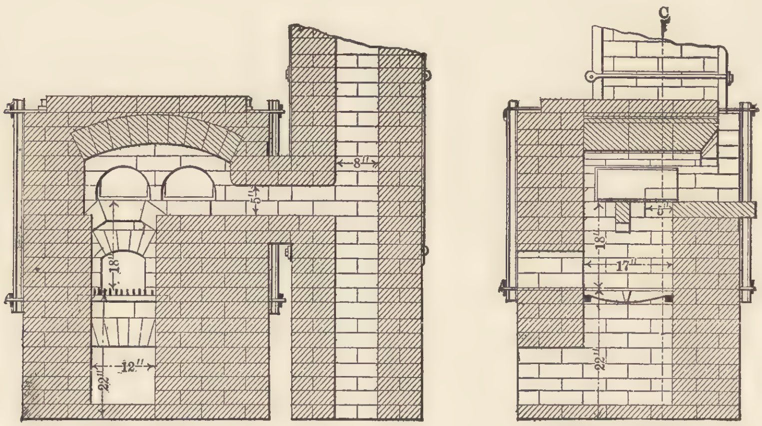
are to be made, and the operations of retorting amalgam and smelting of bullion to be carried on. The assay office consists of assay room and private room. The former, 16x18 ft., contains a bucking-board, a mortar-block, a work-table with pulp-balance and shelves, a gasoline-tank and two furnaces. The private room, 16x14 ft., has a stove, a drawing-board, a button balance and a bed. Full details as to construction and materials required are given. Two carpenters erected the office in six days.

Assay Furnaces.—U. B. Hough[3] publishes the accompanying drawing (Fig. 2) of a double two-muffle furnace used by some of the silver-lead mining companies of Washington and Idaho. He advises to follow the

[1] THE MINERAL INDUSTRY, 1904, XIII, 281.
[2] *Mining Reporter*, 1905, LI, 75.
[3] *Engineering and Mining Journal*, 1905, LXXIX, 1138.

dimensions closely. The grate-bars and doors of the furnace are cast iron, the doors are lined with cast-iron plates having 0.5-in. perforations. The furnace is fired with soft coal, of which 175 to 250 lb. are consumed per day, while the number of assays made ranges from 75 to 150. The furnace in daily use is burnt out in 19 to 22 months; the stack built in 1898 has not required any repairing, and seems destined to last still for a long time.

E. C. Woodward[1] has constructed a two-muffle furnace to be heated with coal. As seen in Figs. 3 and 4, the muffles are placed side by side. The left-hand muffle is above the fire-box and will be hotter than the



FIGS. 3 and 4.—TWO-MUFFLE FURNACE.

right-hand, but the latter will be sufficiently heated by the gases which are forced to pass around it.

W. R.[2] advocates placing abreast the muffles of a furnace, each having a separate fireplace and flue, the flues terminating in a single chimney.

Gillio[3] has constructed a portable assay furnace to be heated with coal oil flashing at 150 deg. C. The furnace is a tin can, 12 in. in diameter by 16 in. high, lined with fire-clay to a thickness of 0.5 in. It will take four H-crucibles, which stand on a platform consisting of two pieces separated by a slot 1 in. wide, through which the flame is admitted at the side. The top is partly covered by two tiles so as to force the flame to play around the crucibles. The hottest zone of the furnace is that farthest away from the burner; thus the two remote crucibles run down first. When these have been poured, the two cooler crucibles are put in their places and two newly charged crucibles introduced.

[1] *Mining Reporter*, 1905, LII, 419.
[2] *Engineering and Mining Journal*, 1905, LXXX, 20
[3] *Proc.* Australasian Institute of Mining Engineers, 1905, X, 270; *Engineering and Mining Journal*, 1905, XXX. 152.

A. E. Peterson[1], referring to the data of F. C. Bowman[2] in regard to the consumption of oil in an assay muffle-furnace, gives 0.85 gal. per hour as the amount of gasoline required for firing a Calkins Advance No. 10 combination furnace[3] which has a capacity of twelve 12-gram crucibles and eighteen 1.5-in. cupels at one firing. While this figure exceeds that of Bowman by 0.20 gal., the larger consumption is more than offset by the larger capacity of the furnace. The time required for making 24 fusions and cupellations (12 fusions and 12 cupellations at a time) is 1 hour and 40 minutes, starting with everything cold.

A. M. McDuffee[4] patented an assay muffle in which the bottom is detached from the arch, thus making a single furnace suitable for crucible, scorification and cupellation work. W. W. Case[5] patented a combined crucible and muffle furnace, to be heated by oil or gas.

Assay Implements.—E. Keller[6] describes several labor-saving appliances[7], devised by him and his assistants at the Baltimore copper works. A drawing and description of the new laboratory erected by him in Baltimore furnishes another example[8] of a modern laboratory designed to carry on dry and wet assays. Attention may be called to the new tools which greatly expedite the work of the assayer. These are multiple tongs which enable the operator to handle 20 scorifiers at one time; two sharp-edged shovels of different widths, and a scraper to place cupels in rows in the muffle and to withdraw them; a device for placing in one operation all the lead buttons in the cupels properly alined in the muffle; and a parting water-bath (having a constant water level) with tray capable of holding 42 test-tubes. Keller advocates bringing the water in the bath nearly to a boil before setting into it the tray with test-tubes charged with dilute nitric acid (1:9) and the doré buttons, as thus the gold remains coherent, even if the proportion of silver and gold is as high as 500:1. He thus confirms the practice advocated by Rose (see later).

W. Ainsworth & Sons,[9] Denver, Colo., have put on the market a new portable button-balance of the columnless type, which has a 5-in. beam. The same firm[10] has on its latest button-balances an improved multiple-rider carrier. Each rider is carried by an individual arm which is marked with the weight of its rider. When a weighing is completed, the figures on the arms that are down show the combined weights of the riders on the bar.

The W. A. Oliver Manufacturing Company, of Buffalo, N. Y.,[11] has

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[1]*Engineering and Mining Journal*, 1905, LXXIX, 574.
[2]THE MINERAL INDUSTRY, 1904, XIII, 281.
[3]*Ibid.*, 1902, XI, 431.
[4]United States Patent, No. 797,901, Aug. 22, 1905.
[5]*Ibid.*, No. 798,949, Sept. 5, 1905.
[6]*Trans.* American Institute of Mining Engineers, Feb., 1905.
[7]THE MINERAL INDUSTRY, 1904, XIII, 283.
[8]*Ibid.*, XIII, 281.
[9]*Engineering and Mining Journal*, 1905, LXXIX, 731.
[10]*Ibid.*, LXXIX, 1105.
[11]*Metal Industry*, 1905, III, 51.

brought out a small rolling-mill for silversmiths, which appears well suited for rolling assay buttons of doré silver.

Dry Assay of Lead Ores.—A note[1] states that at the different plants of the American Smelting and Refining Company the roasting of mixed sulphide ores as the first step in assaying has been given up. Further, the addition of fluorspar to the assay charge of Cripple Creek ores is said to give higher results than when this flux is not used.

W. F. Love[2] strongly advocates the assaying of galena in an iron crucible, as with the necessary experience it gives results which are accurate to 0.025 per cent. He fortifies his claim with the fact that with a pure galena he obtained 85.86 and 85.87 per cent. Pb. The foreign matter in the galena was found by analysis to be Fe_2O_3 0.22 per cent. and $CaCO_3$ 0.53 per cent., or a total of 0.75 per cent. The results simply show that with pure galena excellent results can be obtained in an iron crucible, but all assayers will agree that pure galena is not of common occurrence.

O. J. Frost[3] discusses his combination method[4] for assaying galena ores for lead. It consists in adding to a 10-gram ore charge (galena concentrate), in 10 to 15 minutes after complete fusion, 10 to 15 grams of KCN and 3 to 5 grams with pyritous and silicious ore, the temperature being sufficiently low for the charge not to boil over. With a proper heat, the charge will be ready to pour in from 30 to 40 minutes from the time of complete fusion. The buttons are soft and contain, exclusive of the silver and gold content, 99.5 per cent. Pb as long as readily reducible oxides of Cu, Sr, Bi and Sb are absent. Frost has found the following two charges to work satisfactorily: (1) The Cœur d'Alene flux without KCN: sodium bicarbonate, 4, potassium carbonate, 4, powdered borax, 2, flour, 1 part; and (2) the Richard flux with KCN: sodium bicarbonate, 4, potassium carbonate, 4, powdered borax, 2, flour, 1 part. Nails and salt cover are added to suit. He further advocates slow fusion requiring for a charge 45 to 60 minutes, and even more, viz.: low heat 15 to 20 minutes, rising temperature 20 to 30 minutes, final moderate heat. Some results are subjoined:

No.	Flux 1% Pb.	Flux 2% Pb.	Wet % Pb.
1	55.3	55.3	55.66
2	32.5	33.3	33.63
3	62.0	62.4	62.7
4	44.6	44.75	45.36
5	34.75	34.7	....
6	64.9	64.9	66.0
7	19.94	20.47	22.21
8	6.48	7.43	(a) 8.05

Nos. 1–6 galena; 7, pyritic ore; 8, silicious ore. (a) Result too low.

[1] *Engineering and Mining Journal*, 1905, LXXX, 692.
[2] *Journal* Society Chemical Industry, 1905, XXIV, 6.
[3] *Western Chemist and Metallurgist*, 1905, I, 239.
[4] THE MINERAL INDUSTRY, 1902, XI, 432.

Wet Assay of Lead Ore.—G. A. Guess[1] describes the apparatus and method for the electrolytic determination of lead and copper. The cathode is of platinum foil 0.001 in. thick, which weighs 1.5 grams. It consists of a blade 4 cm. wide by 6.25 cm. long, sand-blasted and corrugated lengthwise, and a tongue 0.7 cm. wide by 6.25 cm. long. The immersion area is 50 sq. cm. The anode, 0.5 cm. wide by 12.5 cm. long, with a median corrugation, is made of the same thickness of platinum foil as the cathode and is also sand-blasted, as it then can carry up to 600 mg. lead peroxide; if not thus roughened, only a very small amount will adhere firmly. One anode flanked by two cathodes is used in a cell. The electrodes are connected to slotted aluminum terminals in which they are held by contact pressure; the terminals are $\frac{5}{16}$ in. rods projecting 2 in. The mode of operating is as follows: Weigh ore into a tall 100-c.c. battery beaker, digest with 10 c.c. nitric acid, boil, dissolve lead sulphate with 10 to 20 c.c. of saturated solution of ammonium nitrate containing 20 c.c. free ammonium, nearly fill beaker with water, add 10 to 20 c.c. nitric acid, and electrolyze with a current of 1.5 to 2 amperes. Within two hours the lead is completely deposited. Remove anode, wash in water and alcohol, ignite, weigh and multiply by 0.855 instead of by the theoretical factor 0.866. The presence of manganese and antimony is made harmless by the use of an excess of nitric acid; bismuth, arsenic and tellurium have to be removed before electrolysis. In weighing out the pulp, Guess uses 855 mg. as a unit to avoid the necessity of calculations. The same electrodes are used for copper determinations. Here the ore is digested with 7 c.c. nitric acid and boiled to expulsion of red fumes; now 2 c.c. "dope" (a deep red solution obtained by boiling No. 4 hard oil of the Standard Oil Company with nitric acid, cooling and removing the grease) diluted with nitric acid is added, the beaker is filled with water, allowed to settle for a moment and the solution electrolyzed with a current of 1.5 amp. for three hours, when all the copper will have been deposited. Arsenic and antimony do not contaminate the deposit, which adheres firmly if the cathode has been sand-blasted and does not fly off upon heating.

Assay of Gold Ores.—A. Whitby[2] believes that in assaying a gold ore the degree of fine-crushing of the sample has a considerable influence upon the result, the finer pulp giving a higher result than the coarser. He gives

Pulp.	30–60	150	60–90	150
A	4.10	4.52	4.20	4.88
B	5.00	5.55	5.00	5.70
C	5.10	5.64	4.40	5.28
Average	4.73	5.23	4.53	5.28

[1]*Trans.* American Institute Mining Engineers, July, 1905.
[2]*Journal* Chemical, Metallurgical and Mining Society of South Africa, 1904, V, 95.

data (see table, p. 653) in pennyweights as example of his experiments, but acknowledges that as yet his researches are incomplete.

R. Marsh[1] commenting upon the paper attributes the higher figures obtained with the finer material to the common quick fusions, which are often not sufficient to decompose all the particles of a charge.

T. K. Rose[2] in discussing cupelling and parting gives the accompanying illustration (Fig. 5) of the assay muffle in use at the Royal Mint, London. The muffle, of the Salamander brand of the Morgan Crucible Company,

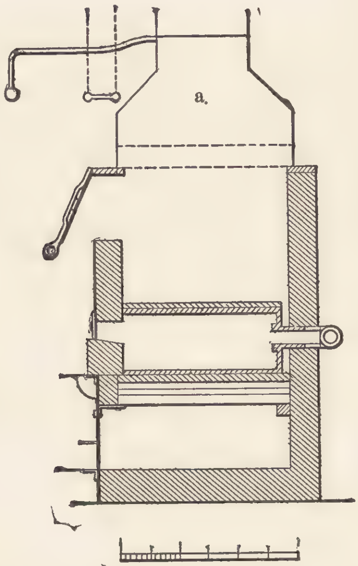

Fig. 5.—Muffle Furnace of Royal Mint, London

is of graphite and does not crack easily. In the first heating, the graphite on the inside burns off and then has no effect upon the assays. The muffle is 6.5 in. wide by 15.5 in. long, and holds 72 small cupels for bullion assays, or 18 large ones for lead buttons from ores. The two striking peculiarities of the muffle are that the air is admitted at the front only, near the top, thus avoiding irregularities in the results, and that at the back the necessary draft is obtained through a graphite pipe, 1.5 in. in diameter, ending outside of the furnace in an iron tube (with damper) leading into the flue. Given the melting point of lead at 326 deg. C., the uncovering temperature at about 675 deg. C., and the freezing point of litharge at about 640 deg. C., a good cupelling temperature is about 700 deg. C. This is too high for forming feather-litharge, nevertheless the losses

[1] *Mining Reporter*, 1905, LI, 190.
[2] *Journal*, Chemical, Metallurgical and Mining Society of South Africa, 1905, V, 165, 212, 237, 311; *Engineering and Mining Journal*, 1905, XXXIX, 708.

are low. In the presence of impurities, such as iron, antimony, etc., the temperature will have to be raised to 900 deg. C. This, however, increases losses, as shown in the following table:

Charge.	Loss at 700° C.	Loss at 900° C.
Gold, 0.001 gm. Silver, 0.006 " Lead, 25 "	Gold, 0.45%	Gold, 1.05%
Gold, 0.001 gm. Silver, 0.010 " Lead, 25 "	Gold, 0.39%	Gold, 0.80%

The beads set at 900 deg. C., while their melting points were found to be above 900 deg. C. The results show, as was already known, that the loss of gold decreases with an increase of silver. Drawing attention to the influence of the cupel upon the loss of gold, Rose compared magnesia with English and French (Deleuil) cupels.

Loss of Gold at 900° C.	Grams. Au, 0.001 Ag, 0.004 Pb, 25	Grams. Au, 0.001 Ag, 0.008 Pb,10	Grams. Au, 0.5 Ag, 1.25 Pb, 10
Magnesia cupel	1.9	1.2	1.8
Bone ash cupel, English	1.4	0.9	5.2
Bone ash cupel, French	(a)0.06	(a)0.055	n. d.

(a) Real losses; apparent gains were 0.02 and 0.025 respectively.

The results given in the above table show the loss in the magnesia cupels to be greater than in those made of bone-ash, and that the Deleuil cupels are not as regular as they are often believed to be. In order to test the effects of impurities upon losses, Rose made up charges of 1 mg. gold, 4 mg. silver, 1 gram impurity and 25 grams lead, and cupelled these at 1000 deg. C. The results are given in the subjoined table:

Impurity.	Loss of Gold. %	Loss of Silver. %	Remarks.	Impurity.	Loss of Gold. %	Loss of Silver. %	Remarks.
None	1.2	11.8	....	Mo	11.0	26.2	No Scoria.
Sn	2.0	13.9	Slight Scoria.	Va	7.7	21.7	No "
As	3.9	16.3	Much "	Cu	10.0	32.6	No "
Sb	5.3	13.2	No "	Bi	21.8	27.9	No "
Zn	9.3	17.6	Much "	Th	23.1	34.4	No "
Cd	3.5	13.1	Ring of "	Te	55.8	67.9	No "
Fe	4.0	16.6	No "	Se	54.1	64.5	No "
Mn	13.6	24.3	Some "				

In regard to the parting, Rose recommends the following procedure: Fill a porcelain crucible nearly full with nitric acid of sp. gr. 1.25 (4 acid: 3 water), cover with watch-glass, raise to boiling and drop in flattened bead. The parting is rapid and complete, and the bead hardly ever breaks up, whatever its composition (see Keller, above). Even when it does

the particles are not small (float-gold) and are easily washed. The results with varying ratios of gold and silver are given below:

Charge.			Ratio.	Weight Parted,	Charge.			Ratio.	Weight Parted,
Au, Mg.	Au, Ag.	Pb., Grams.	Au: Ag.	Au, Mg.	Au, Mg.	Ag, Mg.	Pb, Grams.	Au: Ag.	Au, Mg.
1.085	6.4	2	1.9	1.08	0.97	8.1	2	1.3	0.66
1.125	5.85	2	1.2	1.12	0.87	8.0	2	1.2	0.87
9.9	39.9	2	1.0	9.89	0.065	2.21	2	1.4	0.065
10.265	40.35	2	1.9	10.26					

Referring to the above paper, H. R. Edmands[1] gives results of some of his work in cupelling and parting. As to the cavity of the cupel, he prefers one that is shallow to one that is deep, as it offers with a given amount of lead a larger surface for oxidation. Thus, shallow forms cupelled from 10 to 20 per cent. more quickly than deep ones. Comparative tests for which tabulated data are given show that cupels made of equal parts of bone-ash and ''mabor'' (a refractory material composed mainly of magnesium borate) give a smaller loss in silver than when of bone-ash alone, the average losses being 1.19 and 1.64 per cent. In cupelling at a temperature above that for normal work, the average losses were 2.46 per cent. with bone-ash, 2.15 per cent. with morganite, 2.09 per cent. with bone-ash-mabor, and 1.86 per cent. with mabor cupels. In parting low-grade beads, Edmands prefers sulphuric acid as a solvent to nitric acid, as the gold does not break up.

F. K. Rose[2] criticizes the statements of Edmands that shallow cupels are preferable to those that are deep, because Edmands has not considered the loss by absorption, which is greater in the shallow cupel. For this reason the Royal Mint, London, uses deep cupels, and recently increased the depth beyond that which has been customary. Rose does not wish his data, in the preceding abstract, to be interpreted as stating that London-made cupels of bone-ash are better than Deleuil's, or than magnesia cupels. He merely wants to draw attention to the fact that in ore-assays, or with beads weighing 3 mg. or less, the cupel absorption is too great to be neglected.

A. T. French[3] found that, in using cupels 0.25 instead of 0.5 in. deep, the lead cupelled more quickly, but the silver sprouted a great deal more than in 0.5-in. cupels.

W. F. Hillebrand and E. T. Allen[4] have published a paper entitled "Comparison of Wet and Crucible-Fire Methods for the Assay of Gold Telluride Ores, with Notes on the Errors occurring in the Operations of Fire-Assay and Parting." The two ores used for the experiments were

[1]*Engineering and Mining Journal*, 1905, LXXX, 245.
[2]*Ibid.*, 1905, LXXX, 934.
[3]*Ibid.*, 1905, LXXX, 694.
[4]*Bulletin*, No. 253, Series E, Chemistry and Physics, 44; U. S. Geol. Survey, Washington, 1905.

tellurides from Cripple Creek, Colo., assaying 15 and 19 oz. per ton and containing, beside the granite, small amounts of pyrite and calcite. Wet analysis gave the following metallic constituents:

Sample.	Fe	Au	Ag	Cu	Mo
a................	0.0742	0.0506	0.0075	0.0059	0.0015
b................	0.092	0.060	0.0103	0.0070	0.0018

The best dry-assay results were obtained with a crucible charge: ore 1 a. t., sodium bicarbonate 1 a. t., litharge 6 a. t., glass-borax 10 grams, salt cover. It was found that the amount of gold retained by the slag was sufficiently small to be negligible if a suitable charge was used. The loss in cupelling, however, was important, especially that due to cupel absorption, which increased with the temperature, and was found to be greater with pure gold and alloys poor in silver than with alloys rich in silver. The loss by volatilization was generally small, and could be neglected as long as the cupelling temperature was sufficiently low for abundant feather litharge to form, being perhaps compensated by the traces of lead retained by the button. The authors found that gold did not need a higher temperature for brightening than silver, also that gold buttons left in the muffle after brightening did not decrease in weight, on the contrary, they showed a tendency to a slight increase. In parting, they found silver could be completely extracted by nitric acid from inquarted alloys, but required more than two boilings and a subsequent washing. They did not find that nitrous acids added to nitric acid had any solvent effect upon gold, and re-established the fact that in ore assays the losses in parting with pure nitric acid are negligible, if they occur at all.

J. C. Bailar[1] calls attention to the difficulty of obtaining correct values in assaying telluride gold ores, due, to a large extent, to cupel absorption. Thus, in cupelling a specimen of pure telluride of gold and silver, wrapped in sheet lead, no button at all was obtained; in a second case the first button weighed 136 mg., assaying the cupel gave 259 mg. doré silver, and assaying the second cupel 19 mg. Other examples of cupel absorption are given.

E. C. Woodward[2] made tests to determine the slagging and cupelling loss in assaying telluride gold ores. An average of eight tests, using with one exception 0.5 a. t. ore and fusing 30 to 120 minutes, gave assay button 95.27, cupel button 0.225 and slag button 37.2, or a total of 96.11 mg.; i. e. cupelling loss 0.23 per cent. and slagging loss 0.38 per cent. The slagging loss is high, as 0.5 a. t. ore was used. The yield in precious metal decreased inversely with the time given to the fusion.

[1] *Western Chemist and Metallurgist*, 1905, I. 119.
[2] *Ibid.*, I. 120.

C. F. Eaton[1] advocates determining the loss in gold through cupel absorption by crushing the soaked part of the cupel to moderate fineness, boiling it in a Florence flask in dilute nitric acid (1:5) in the presence of a globule of mercury (about 1 gram) which is to collect the gold, and then removing the gold from the amalgam by means of nitric acid. The correction for silver is to be made by increasing its amount in the proper ratio of the two metals. The following table gives some of the results obtained:

Cupel No.	Cupel Size. Grams.	Gold in Cupel Button.	Gold Recovered from Cupel.	Cupel No.	Cupel Size. Grams.	Gold in Cupel button.	Gold Recovered from Cupel.
1	30	0.00	0.72	7	30	30.59	2.41
2	30	1.65	0.49	8	30	0.00	1.50
3	20	2.89	0.64	9	30	0.00	trace
4	20	0.96	0.17	10	20	7.56	6.64
5	20	2.24	0.74	11	30	19.00	2.52
6	20	0.78	1.60	12	30	0.00	trace

Assaying Matte for Gold and Silver.—G. Crerar[2] gives the following method for making up the charges of a crucible in assaying mattes and furnace products for gold and silver: Fill eight 20-gram Battersea crucibles two-thirds with litharge (about 0.8 a. t.), place in each 0.25 a. t. matte, pour charge on rubber sheet (18x18in.), roll, return to crucible, cover with a little litharge, then with 0.5 a. t. borax, and lastly with 0.5 a. t. salt. Fuse in hot muffle until reactions cease (in about 25 minutes), pour into heavy iron molds, place eight 3-in. Bartlett-shaped scorifiers in a muffle, break crucible charges, place buttons with some borax in scorifiers which are now sufficiently hot for the work. When covered, combine charges of two scorifiers, pour and cupel.

With iron sows, place drillings in bottom of crucible and proceed as above. Blister copper or bottoms rich in copper are preferably first matted by heating with 10 to 15 grams of sulphur before the litharge is placed on top.

Assay of Copper Matte and Copper Bullion.—H. H. Atkins[3] describes the methods in use at the Philadelphia works, Pueblo, Colo., for determining gold, silver, lead and copper in blister copper and rich copper matte.

Blister copper: charge 10 grams in 12-oz. beaker, add 90 c.c. water, 25 c.c. concentrated nitric acid and cover beaker, give additional 25 c.c. nitric acid, and when action has ceased add four drops concentrated hydrochloric acid; when copper is all dissolved, boil, filter off coagulated silver chloride, collect filtrate in 2-liter flask and fill to mark. Take duplicate samples of 100 c.c., transfer to beaker, add 75 to 100 c.c. almost saturated ammonium nitrate, and electrolyze. Gold: Make 10 scorification assays in 3-in. scorifiers with charge, ore 0.1. a. t., test lead 70 grams, silica 1 gram, borax glass 1 gram, and test-lead cover. Pour slag twice from all scorifiers, and

[1] *Mining Reporter*, 1905, LI. 500.
[2] *Western Chemist and Metallurgist*, 1905, I. 229.
[3] *Bulletin*, Colorado School of Mines, 1905. II, 14

combine two and three scorifiers when covered, pour into molds, place the buttons from 0.2 and 0.3 a. t. ore into a scorifier, add 50 grams test lead and some borax, scorify to 30 grams button, cupel and part. A quicker method, but not so accurate, is to do away with scorification; viz., place twenty $\frac{1}{20}$ a. t. ore charges in 20 large cupels, cover with 40 grams test lead, introduce into hot muffle and open quickly, placing some soft wood in front of the cupels. The following assays of nine lots in ounces per ton are instructive:

Lot No.	1	2	3	4	5	6	7	8	9
Scorifier 1	0.52	0.63	0.77	2.25	1.99	1.86	0.60	1.30	1.82
Scorifier 2	1.54	0.61	0.75	2.23	1.97	1.84	0.57	1.28	1.84
Direct cupellation	0.60	0.62	0.76	2.26	1.87	1.80	0.58	1.36	1.87

Silver: Weigh four portions of 0.5 a.t. into 18-oz. beakers, dissolve as with copper, add five drops concentrated hydrochloric acid for ore assaying 400 oz. or less per ton, and one drop more for each additional ounce, stir vigorously with glass rod and allow to stand over night (boiling hastens settling, but is not safe). Filter through double filter, wash two to three times, burn filter in 2-in. scorifier, add 25 grams test lead and pinch of borax glass, scorify until half-covered, pour the 15-gram button, cupel and weigh.

Copper Matte: Lead and copper—make triplicate assay. Dissolve one gram in 15 c.c. nitric acid, add a little hydrochloric acid, boil until nitrous fumes are driven off, add 25 c.c. dilute sulphuric acid (1:1), heat to white fumes, cool, add 50 c.c. cold water, boil, filter and wash. Dissolve lead sulphate in ammonium acetate, boil and titrate with ammonium molybdate. Boil filtrate with aluminum foil, filter precipitated copper, transfer aluminum into filtrate and boil for 20 minutes to precipitate last traces of copper (or remove traces of copper from filtrate with hydrogen sulphide and dissolve copper sulphide with bromine water). Wash precipitated copper from filter with little water into beaker, decant first water through second filter and filter second precipitate through it; add 5 c.c. nitric acid to beaker containing aluminum to dissolve adhering copper, pour acid upon second precipitate and collect in beaker containing first precipitate. Add to solution a couple of crystals of potassium chlorate to precipitate arsenic, evaporate to syrupy consistency, add 50 c.c. water, 6 c.c. ammonium nitrate, boil, neutralize with acetic acid, boil for a few minutes, allow to get cold, add 5 grams potassium iodide and titrate with standard sodium hyposulphite solution.

Gold: A crucible assay is made. With matte containing less than 50 per cent. Cu, take 20-gram crucible, weigh 0 .25 a.t. matte, mix with 25 gram non-reducing flux (sodium bicarbonate 4, potassium carbonate 4, powdered borax 1), 10 grams borax glass, 5 grams silica and 80 grams

litharge, and cover with flux; start with moderate heat, increase temperature gradually, keep at high heat for 20 minutes and pour when quiet. The button weighs 25 to 30 grams. With matte assaying 50 to 75 per cent. Cu reduce the amount of matte in charge. With matte assaying 75 per cent. Cu, use 0.1 a.t. matte, 25 grams reducing flux (same as above, but add 1 part flour), 10 grams borax glass, 5 grams silica and 80 grams litharge, and proceed as above.

Silver: The method is similar to that of blister copper. Weigh 4 portions of 0.5 a.t. into 18-oz. beakers, add 25 c.c. water, stir add, 25 c.c. concentrated nitric acid, wash down sides of beaker, add 30 c.c. nitric acid, cover, put in warm place until copper is dissolved (about two hours), add five drops hydrochloric acid, 5 to 6 c.c. diluted sulphuric acid, boil two to three minutes, wash sides of beaker with 100 c.c. water, stir vigorously, allow to stand for half an hour; then filter through 12.5 to 15 c.c. paper, wash three or four times, transfer filter to 20-gram crucible, burn filter in muffle, keeping crucible there until free sulphur is burning well, remove and let sulphur burn off completely. Flux (without mixing) with 20-gram reducing flux, a pinch of borax glass, 50 grams litharge, cover with non-reducing flux, start fusion with low heat, and finish hot. Time required, 45 minutes.

Determination of Copper, Arsenic and Antimony in Base Bullion.—H. C. Parmelee[1] describes the method of determining copper, arsenic, and antimony as carried out at smelters in a sample representing the average of a month's shipment. Sample: From the usual sample bar, 4x12x0.5 in., samples weighing over 1 a.t. are punched for the reduction of gold and silver, the clippings which the assayer cuts off in trimming a sample down to 1 a.t. are reserved and cast into a bar at the end of the month; punch samples of this bar furnish the material for analysis. Analysis: Dissolve 10 grams in No. 3 beaker with 50 c.c. diluted nitric acid (1:4) at a quick heat, evaporate to incipient crystallization, add 25 c.c. hot water, boil and filter hot. Transfer the residue (insoluble antimony oxides) to a beaker with 10 c.c. water, add 2 c.c. hydrochloric acid and 1 c.c. nitric acid, warm until dissolved, filter through original paper, wash with hot water, evaporate to dryness (avoiding much excess over 100 deg. C.), dissolve in tartaric acid (1 gram in 10 c.c. water) and a few drops hydrochloric acid, boil and add to main solution (see below). Add to original solution (150 c.c. or less) enough hydrochloric and sulphuric acid to precipitate silver and lead, and follow closely with 50 c.c. alcohol, stirring rapidly; allow to stand three hours, filter through Hirsch funnel or Witte plate, wash by decantation three times with solution of alcohol and hydrochloric acid (4:1), transfer precipitate to filter and

[1] *Western Chemist and Metallurgist*, 1905, I, 60; *Mining and Scientific Press*, 1905, XC, 340; *Mining Reporter*, 1905, LI, 238.

pump dry. Boil alcohol from solution, reduce volume to 100 to 125 c.c., add antimony solution from above, make strongly alkaline with ammonia, boil a few minutes to expel excess ammonia, add 2 c.c. hydrochloric acid and introduce hydrogen sulphide for 45 minutes. Filter, wash, transfer precipitate to beaker with minimum of water, add 10 c.c. of 2-gram caustic potash solution, heat until arsenic and antimony are dissolved, allow to stand 30 to 40 minutes in warm place, filter through original filter and wash, keeping filtrate to 40 c.c. Add to solution one gram of potassium chlorate and 50 c.c. hydrochloric acid, boil clear, filter, make ammoniacal, add one-third volume of solution of ammonia, filtering off any precipitate that may form, precipitate with magnesia mixture and determine arsenic in the usual way. Boil filtrate from arsenic, to expel ammonia, add 25 c.c. hydrochloric acid, cool, add excess potassium iodide and titrate with sodium hyposulphite. Determine copper insoluble in caustic potash by stirring in nitric acid and proceeding by the iodide method.

Assaying Bars of Metal for Gold and Silver.—W. E. Burlingame[1] describes at length the methods of assaying bars of base bullion, copper bullion, doré bullion, gold bullion and melted jewelry, for gold and silver in the well-known Denver assay office, and the methods employed for melting and refining bars containing much base metal.

T. K. Rose[2] discusses briefly the changes that have been made during the last three years at the Royal Mint, London, in the assay of gold bullion. For a general description of the mode of operating he refers to his well-known treatise "Metallurgy of Gold." In cupelling, 72 cupels are placed on a plumbago tray, 11.5 in. long by 6 in. wide, which has raised edges to keep the cupels in place and two grooves along the bottom to receive the flat prongs (11.5 in. long by 1.25 in. wide) of the fork, by means of which the tray is lifted in and out of the furnace. The tray can be used 20 to 30 times. The 72 cupels are changed simultaneously by means of a charging-tray, devised by A. Westwood in 1893, which resembles very much that of Keller referred to above. The draft in the muffle is regulated as shown in Fig. 5; the muffle is long enough to allow placing the first row of cupels 7 in. from the front. The cupels are deep to promote regularity of work; i.e., the cavity is 0.75 in. in diameter and 0.3 in. deep. A charge of 4 grams lead is cupelled in 10 to 15 minutes. In parting the ratio Ag 2:Au 1 has been adopted, replacing that of Ag 2.75: Au 1 formerly employed, as with the latter the attack by the acid was violent and caused fragments to be detached from the cornets.

Assaying in a German Smelting Plant.—H. Nissenson[3] describes the

[1] *Bulletin*, Colorado School of Mines, 1905, II, 62.
[2] *Journal*, Chemical, Metallurgical and Mining Society of South Africa, 1905, VI, 367: *Engineering and Mining Journal*, 1905, LXXX, 492.
[3] *Metallurgie*, 1905, II, 430.

methods at the Stolberg-Westphalia works. *Lead ore:* Assays are made in iron crucibles; the flux consists of 7 parts by weight of sodium carbonate (98 per cent. pure), 6 borax and 1 cream of tartar. Ore, 25 grams, mixed with half a soupspoon of flux, is charged into a heated crucible and covered with 100 grams flux. When fused, the draft is shut off, the crucible kept in the furnace until quiet fusion is reached, and then poured. *Silver ore:* Assay in iron crucible; ore 10 grams, litharge 25 grams; flux as with lead ore. Scorification assay with ores containing 29 oz. silver per ton and over. *Leady ore:* 5 grams, mixed with 10 grams assay lead, covered with 25 grams lead and pinch of borax added; sometimes 1 gram iron is added to the charge. *Dry Silver ore*: 2.5 grams, mixed with 10 grams assay lead, covered with 20 grams lead, and pinch of borax added. *Gold ore:* Ore 25 grams, mixed with 25 to 30 grams litharge and fluxed as is lead ore. *Prepared Jeweler Sweep:* Sweep 25 grams, mixed with 20 grams special flux (10 grams black flux, 40 grams sodium carbonate, 10 grams borax), covered with 40 grams special flux, salt cover added. *Cupellation:* Lead buttons are scorified with 12.5 to 40 grams assay lead; cupels are of bone-ash or neat cement. *Copper and copper alloys:* Dissolve 25 grams in diluted nitric acid, add 25 grams litharge or red lead to filtrate, precipitate silver with salt, add some sulphuric acid, settle and filter; combine the two residues and assay as one would a lead ore. *Tin:* Dissolve 25 to 100 grams in diluted sulphuric acid, filter, mix residue with lead and cupel. *Nadorite* (lead ore containing antimonic oxide): Ore (Pb 44 per cent., Sb 22 per cent.) 10 grams, mixed with 25 grams litharge, fused in iron crucible; slag is fused again with 15 grams litharge; the two lead buttons are scorified together and cupelled. *Residue from zinc retorts:* Roast in muffle and assay as you would jeweler sweep.

AUSTRALASIA

In the following tables the production of minerals and metals in each of the Australian States (except South Australia) and New Zealand is separately itemized. In the tables relating to foreign commerce, however, the States are not separately treated, the combined statistics of the Commonwealth now being officially reported.

MINERAL PRODUCTION OF NEW SOUTH WALES. (a)
(In metric tons or dollars; £1=$5) (b)

Year.	Alunite.	Antimony and Ore.	Bismuth Ore.	Chrome Ore.	Coal.	Coke.	Cobalt Ore.	Copper Ore.	Copper Matte, Ingot and Regulus.
1895	845	486	3	4,297	3,798,406	28,072	26	55	3,743
1896	1,394	134	42	3,914	3,972,069	26,774	...	15	4,464
1897	736	172	3	3,433	4,453,729	65,229	...	169	6,458
1898	2,988	83	29	2,145	4,781,551	83,538	119	181	5,577
1899	935	332	16	5,327	4,670,580	98,074	193	445	5,574
1900	1,946	252	11	3,338	5,595,879	128,238	145	867	6,243
1901	3,196	90	21	2,523	6,063,921	130,944	112	655	6,184
1902	3,702	58	10	508	6,037,083	128,902	35	3,190	5,560
1903	2,524	13	23	1,982	6,456,523	163,161	155	1,750	8,094
1904	376	111	41	404	6,116,126	173,742	6	2,470	6,654
1905	2,745	394	56	53	6,738,252	165,568	*nil*	487	7,899

Year.	Diamonds. *Karats.*	Gold. (b)	Iron. (c)	Iron Oxide.	Lead. Argentiferous. (g)		Lead. Pig. (g)	Molybdenite.	Opal.
					Ore.	Metal. (f)			
1895	1,313	$6,403,312	2,441	154	193,236	30,162	201	..	$30,000
1896	8,000	5,222,971	4,797	381	271,641	19,886	24	..	225,000
1897	9,189	5,373,596	3,291	234	275,249	18,395	32	..	375,000
1898	16,493	5,847,680	5,283	398	394,676	10,270	1,745	..	400,000
1899	25,874	7,899,075	6,604	402	431,126	20,614	4,896	..	675,000
1900	9,828	5,211,097	7,861	318	426,480	19,400	6,807	..	400,000
1901	9,322	3,587,040	10,591	131	406,560	17,191	3,394	..	600,000
1902	11,995	3,333,064	6,099	191	371,496	15,660	4,685	16	700,000
1903	12,239	5,255,421	3,183	1,212	335,870	18,779	3,561	31	500,000
1904	14,296	5,576,966	6,404	421	373,362	30,212	5,977	26	285,000
1905	6,354	5,669,099	4,518	547	420,266	28,244	214	20	295,000

Year.	Platinum. *Kg.*	Shale Oil.	Silver—*Kg.* (g)	Stone. Limestone Flux.	Tin. Ore.	Tin. Block.	Tungsten Ore.	Zinc. (d) (g)
1895	12.9	60,377	17,112	105,861	77	1,346	..	......
1896	75.8	32,348	6,307	90,347	98	1,147	..	......
1897	61.2	34,635	4,666	68,671	14	799	..	29,303
1898	38.9	30,164	16,580	9,401	1	639	..	39,564
1899	19.8	37,307	21,525	1,016	5	749	..	50,677
1900	15.6	23,229	24,080	17,273	15	1,087	..	20,594
1901	12.1	55,650	13,950	26,995	11	659	..	642
1902	11.6	63,886	33,195	17,630	23	502	..	1,281
1903	16.5	35,332	34,195	24,205	556	949	9	21,086
1904	16.6	38,477	34,880	25,374	586	1,084	106	58,523
1905	12.4	38,838	12,987	15,180	726	817	228	105,189

(a) From the Annual Report of the Department of Mines, New South Wales. (b) Where gold is reported £1=$4.866. (c) Manufactured from old iron. (d) Spelter and concentrate. (e) Includes minor quantities of lead carbonate and chloride, the product of the leaching plant at Broken Hill. (f) Includes a small quantity of silver-sulphide. (g) Exported.

MINERAL PRODUCTION OF NEW ZEALAND. (*a*) (*b*)
(In metric tons or dollars; £1=$5.) (*c*)

Year.	Antimony Ore.	Chrome Ore.	Coal.	Coke.	Copper Ore.	Gold. (*c*)	Kauri-gum.	Manganese Ore.	Silver—*Kg.*
1895	55	....	738,280	293	..	$5,655,090	7,544	213	2,644.6
1896	21	....	805,537	107	..	5,067,589	7,240	66	2,933.3
1897	10	....	854,164	..	..	4,769,673	6,748	183	5,719.8
1898	..	....	921,546	9	2	5,258,642	10,063	220	9,140.0
1899	..	....	990,838	18	..	7,363,100	11,294	137	10,865.6
1900	3	28	1,111,860	..	12	7,005,103	10,322	166	10,202.0
1901	30	....	1,259,521	..	3	8,533,908	7,662	211	17,762.0
1902	..	128	1,386,881	..	..	9,495,673	7,549	...	20,970.3
1903	..	....	1,542,953	..	6	9,916,086	9,507	71	28,364.3
1904	..	....	1,562,443	..	..	9,671,180	9,350	199	34,042.3

(*a*) From New Zealand Mines Statement, by the Hon. James McGowan, Minister of Mines, Wellington, 1905. (*b*) The exports are stated to be identical with the production, with the exception of coal, the exports of which were as follows: In 1895, 87,363 tons; in 1896, 80,796 tons; in 1897, 77,280 tons; in 1898, 57,333 tons; in 1899, 90,912 tons; in 1900, 116,216 tons; in 1901, 162,197 tons; in 1902, 191,696 tons; in 1903, 154,769 tons; in 1904, 167,864 tons. (*c*) Where gold is reported £1=$4.866.

MINERAL PRODUCTION OF QUEENSLAND. (*a*)
(In metric tons or dollars; £1=$5.)

Year.	Bismuth Ore.	Coal.	Copper Ore.	Gems other than Opal.	Gold—*Kg.* (*d*)	Lead.	Manganese Ore.
1895	60	328,237	441	(*b*)$29,575	$13,056,414	369	361
1896	..	377,332	589	(*c*)	13,235,842	628	305
1897	1	364,142	293	(*c*)	16,699,477	391	403
1898	8	414,461	63	(*c*)	19,016,763	252	68
1899	2	501,913	164	(*c*)	19,571,662	57	747
1900	8	505,252	386	4,500	20,002,290	207	77
1901	20	548,104	3,110	30,000	12,367,276	570	221
1902	1	509,579	3,845	25,000	13,238,500	271	4,674
1903	11	515,950	4,995	35,000	13,818,653	3,856	1,341
1904	20	520,232	4,440	52,875	13,210,869	2,079	843

Year.	Molybdenite.	Opal.	Silver—*Kg.*	Stone, Building. (*b*).	Tin Ore.	Tungsten Ore.
1895	....	$163,750	6,999	52,206	2,148	25
1896	....	116,500	8,687	(*c*)	1,579	3
1897	....	51,250	7,280	(*c*)	1,222	13
1898	....	33,225	3,235	(*c*)	1,041	79
1899	....	45,000	4,521	164,939	1,322	263
1900	....	37,500	3,514	152,484	1,133	193
1901	....	37,000	17,777	(*c*)	1,638	73
1902	(*e*) 42	35,000	21,813	139,338	2,118	56
1903	(*e*) 24	36,500	19,972	107,780	3,768	200
1904	(*e*) 22	17,750	20,370	72,841	3,986	1,564

(*a*) From *Annual Reports of the Under Secretary of Mines*, Queensland, when not otherwise stated. (*b*) From *Mineral Statistics of the United Kingdom.* (*c*) Not reported. (*d*) Where gold values are reported, £1=$4.866. (*e*) Includes bismuth and tungsten.

MINERAL PRODUCTION OF TASMANIA (*a*)
(In metric tons or dollars; £1 = $5.)

Year	Coal.	Copper Ore and Matte.	Gold. (*e*)	Iron Ore.	Lead-Silver Ore.	Stone.			Tin.
						Limestone.	Freestone, Flagstone, and Building Stones. *Cubic Feet.*	Rubble or Metal.	
1895	33,703	34	$1,033,193	(*f*)	18,194	1,419	21,470	6,200	3,926
1896	44,286	52	1,156,035	203	21,150	2,621	13,575	(*g*)4,556	3,867
1897	43,210	113,261	1,407,447	999	17,806	1,702	82,197	13,274	3,282
1898	49,902	(*b*)	1,369,706	1,296	196,707	45,324	19,560	70,701	2,882
1899	43,803	(*c*)60,985	1,593,834	6,726	424,552	71,747	14,400	12,060	3,333
1900	51,549	(*d*) 4,221	1,538,727	5,141	453,579	47,671	51,509	4,291	2,693
1901	49,963	(*d*)11,401	1,436,326	1,422	804,463	26,545	37,579	244,721	2,516
1902	49,647	(*d*) 8,630	1,467,454	2,424	47,226	(*f*)	(*f*)	(*f*)	1,989
1903	49,856	(*d*) 3,891	1,237,925	6,076	43,103	(*f*)	(*f*)	(*f*)	2,414
1904	62,090	8,826	1,362,587	6,950	51,959	(*f*)	(*f*)	(*f*)	2,104

(*a*) From *Statistics of the Colony of Tasmania.* (*b*) Included with lead-silver ore. (*c*) In addition there were produced 43 tons of copper bullion. (*d*) In 1900 there were produced 9343 tons of blister copper; in 1901, 10,141 tons; in 1902, 7,869 tons and in 1903, 6,791 tons. (*e*) Where gold values are reported 1£=$4.866. (*f*) Not reported. (*g*) Represents cart-loads.

MINERAL PRODUCTION OF WESTERN AUSTRALIA. (*a*)

(In metric tons or dollars; £1=$5.) (*b*)

Year.	Antimony.	Coal.	Copper Ore.	Gold. (*b*) (*c*)	Iron Ore.	Lead Ore.	Limestone.	Silver. *Kg.*	Tin Ore.
1899	..	55,208	3,012	$30,396,596	13,058	84	17,774	.....	340
1900	..	120,305	6,282	29,233,035	12,448	272	16,183	894	836
1901	..	119,721	10,319	35,208,687	20,898	(d)21	18,501	1,893	746
1902	..	143,145	2,298	38,673,323	4,877	(d)36	5,162	2,590	630
1903..........	22	135,568	20,854	42,678,319	224	....	1,301	5,229	830
1904..........	..	140,773	4,033	40,992,284	1,465	....	13,612	12,416	869

(*a*) From the *Report of the Department of Mines of Western Australia.*—NOTE. Previous to 1899 mineral statistics other than for gold were not collected. (*b*) £1=$4.866. (*c*) The value of gold produced in 1895 was $4,280,855; in 1896, $5,200,821; in 1897, $12,481,176; in 1898, $19,418,735. (*d*) Silver-lead ore.

MINERAL PRODUCTION OF VICTORIA. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Coal.	Lignite.	Gold. (*c*)	Stone, Building, etc.	Tin Ore.
1895...........	197,334	1,988	$15,297,276	$450	76
1896.	230,187	5,908	16,640,997	485	47
1897...........	240,057	4,894	16,799,824	(*e*)125,000	48
1898...........	246,845	2,915	17,305,547	100,000	87
1899...........	266,578	(*b*)	17,662,410	(*b*)	158
1900	215,052	(*b*)	16,767,261	175,000	71
1901	212,678	152	16,320,029	225,000	78
1902	228,777	(*b*)	14,899,876	266,975	10
1903..........	65,230	5,752	15,860,815	213,245	34
1904.........	123,695	*nil.*	15,824,952	1,488,075	72

(*a*) From Annual Reports of the Secretary for Mines of the Colony. (*b*) Not reported. (*c*) Where gold values are reported, £1=$4.866. (*e*) Estimated value.

MINERAL IMPORTS OF AUSTRALIA. (*a*).

(In metric tons or dollars; £1=$5.) (*b*).

Year.	Alkalies.	Brass Manufactures.	Bricks, Fire and Glazed.	Britannia, Yellow metal, Etc.	Cement.	Chemicals.	Chinaware and Earthenware.	Coal.	Coke.
1900...	$395,870	$322,230	$25,890	$217,765	53,625	$4,457,060	$1,347,070	7,714	44,169
1901...	418,955	336,170	49,755	142,240	64,519	4,495,020	1,616,645	10,141	36,814
1902...	401,225	218,980	50,115	162,510	48,729	4,237,505	1,358,275	5,149	9,846
1903...	427,540	99,600	22,645	135,425	43,292	3,926,610	940,170	389	4,294
1904...	450,085	72,425	35,525	120,110	25,452	4,126,245	1,151,520	398	4,270

Year.	Copper.		Glass and Glassware.	Gold. (*b*)				
	Ore.	Manufactures.		Ore.	Bullion.	Specie.	Foil. (*c*)	Total Value.
1900....	1,423	$249,735	$1,900,875	$ 14,880	$4,556,007	$78,888	$51,224	$4,700,999
1901....	659	294,870	1,807,805	37,473	3,709,848	18,053	34,704	3,800,078
1902....	1,326	335,105	1,508,530	2,375,513	3,834,510	505,899	30,028	6,740,950
1903....	....	338,915	1,257,860	66,908	5,935,800	6,530	38,680	6,047,918
1904....	....	325,560	1,398,735	68,309	5,684.164	6,297	43,215	5,801,985

Year.	Graphite.	Iron and Steel.						
		Bars, Rods, Girders, Sheets, etc.	Galvanized Plates and Sheets.	Pig and Scrap.	Tin Plate.	Manufactures.	Pipe and Tubes.	Railway Material.
1900	137	100,849	44,598	44,683	$1,520,585	$5,182,730	$1,500,850	$3,888,325
1901	155	94,395	41,075	33,220	1,262,720	7,865,560	1,284,830	6,395,705
1902	121	....	50,099	34,772	1,393,045	7,967,025	1,005,975	6,318,475
1903	252	54,940	40,207	44,898	805,810	3,802,190	1,193,540	2,472,940
1904	193	63,482	46,614	40,063	879,755	2,526,750	1,130,185	920,180

(*a*) From Trade and Customs Returns, Commonwealth of Australia, 1900.—Note. Previous to 1900, each Colony reported its own imports and exports. (*b*) Where gold, silver or platinum values are reported, £1=$4.866. (*c*) Includes a small quantity of silver foil.

Year.	Jewelry and Precious Stones.	Lead Mfrs.	Paints and Colors.	Petroleum Products.			Potassium Nitrate.	Platinum.
				Kerosene—*gal.*	Naphtha—*gal.*	Paraffin.		
1900........	$1,570,560	368	$1,570,880	11,125,905	48,863	1,275	369	....
1901........	1,990,570	166	1,551,270	20,924,640	114,092	1,040	297	....
1902........	1,756,450	376	1,293,960	10,399,931	116,170	1,913	361	....
1903........	2,024,275	380	1,092,420	15,009,609	127,445	2,163	211	$9,255
1904........	2,234,375	242	1,299,875	14,791,319	277,737	530	354	910

Year.	Quick-silver.	Salt.	Silver. (b)			Stone, including slate and marble.	Sulphur.	Zinc.	
			Ore.	Bullion. *Kg.*	Specie.			Bar and Old.	Spelter.
1900............	63.2	22,061	9	190.4	$1,226,208	$364,705	4,972	$213,800	616
1901............	91.0	25,422	743	14.9	772,020	330,185	4,501	154,555	648
1902............	92.6	25,920	252	13.6	439,186	403,185	7,853	132,725	950
1903............	87.5	14,180	...	14.2	160,111	353,465	8,196	158,940	644
1904............	92.6	16,127	...	39.8	154,534	393,135	11,462	134,665	1,057

MINERAL EXPORTS OF AUSTRALIA. (*a*)

(In metric tons or dollars; £1=$5.) (*c*).

Year.	Alunite.	Bismuth.		Cement.	Chrome Ore.	Coal.	Coke.	Cobalt. Ore.
		Ore.	Metal.					
1900..........	1,737	9	4	....	2,190	1,774,980	6,005	130
1901..........	2,853	45	5	......	1,861	1,750,066	4,465	100
1902..........	2,305	5	6	.....	453	1,687,621	6,080	34
1903..........	2,253	33	9	506	1,769	2,063,016	27,345	139
1904..........	344	87	..	1,193	360	1,637,113	2,771	8

Year.	Copper.				Glass and Glassware.	Gold. (*c*)			
	Ore.	Bullion.	Ingots and Matte.	Mfrs.		Ore.	Bullion.	Specie.	Total Value.
1900	4,108	....	15,886	(*b*)	$28,650	$2,379	$19,604,657	$41,898,304	$61,505,340
1901........	10,505	....	17,643	(*b*)	30,855	65,341	22,416,198	43,233,515	65,715,054
1902	7,490	....	21,075	(*b*)	58,325	1,214,208	20,736,800	41,954,939	63,905,947
1903........	2,792	....	27,949	23	61,360	80,591	29,691,889	53,634,629	83,407,109
1904........	4,086	92.1	24,539	14	44,245	46,894	27,073,767	49,284,833	76 405,494

Year.	Iron and Steel.		Jewelry and Precious Stones.	Lead.				Molybdenum Ore.	Paints and Colors.
	Bars, Rods, Scrap, etc.	Manufactures.		Ore.	Pig and Matte.	Argentiferous.	Manufactures.		
1900...	284	$16,965	$290,675	24	17,200	29,711	998	..	$2,990
1901 ..	(*d*)199	59,150	323,750	110	12,761	30,333	1,025	..	3,100
1902. .	144	29,305	345,195	193	16,590	28,950	790	7	4,870
1903...	(*d*)261	40,290	371,375	2,257	28,744	20,093	1,305	36	12,940
1904 ..	224	57,285	380,345	....	73,754	35,847	932	50	6,725

Year.	Platinum.	Salt.	Shale Oil.	Silver.		Stone, including marble and slate.	Tin.		Zinc.	
				Ore.	Bullion, Ingot and Matte.		Ore	Block.	Bar and Old.	Spelter.
1900	.....	4,575	16,792	72,483	192,327.9	$5,470	309	3,273	$3,440	2,601
1901......	.....	7,109	19,587	73,825	196,135.9	24,920	227	2,727	1,690	78
1902......	.....	10,802	27,896	65,084	189,703.3	35,545	466	2,876	2,980	202
1903......	$5,163	7,057	14,483	72,740	202,730.4	10,130	1,219	3,740	20,465	2,730
1904......	5,207	6,419	8,202	101,379	227,972.1	10,450	1,829	4,511	7,420	14,032

(*a*) From "Trade and Customs Returns," Commonwealth of Australia, 1904.—NOTE. Previous to 1900 each Colony reported its own exports separately. (*b*) Included with ingots and matte. (*c*) Where gold, platinum or silver values are reported £1=$4.866. (*d*) Includes a small quantity of scrap.

AUSTRIA—HUNGARY.

In the following tables the mineral and metal productions of the two Kingdoms are reported separately, together with that of Bosnia and Herzegovina. In the tables relating to foreign commerce, the statistics are those of the combined Empire.

MINERAL AND METALLURGICAL PRODUCTION OF AUSTRIA. (a)
(In metric tons.)

Year.	Alum.	Alum and Pyritic Shale.	Antimony.		Asphaltic Rock.	Bismuth Ore.	Coal.	
			Ore.	Metal.			Bituminous.	Lignitic.
1895	885	5,716	695	296	404	185.0	9,722,679	18,389,147
1896	919	25,184	905	422	390	*Nil*	9,899,522	18,882,537
1897	851	21,585	864	425	300	1.0	10,492,771	20,458,093
1898	1,037	28,914	679	343	643	*Nil*	10,947,522	21,083,361
1899	604	19,879	410	271	2,635	0.3	11,455,139	21,751,794
1900	620	3,004	201	153	887	4.0	10,992,545	21,539,917
1901	442	2,551	126	114	541	16.0	11,738,840	22,473,510
1902	62	2,866	18	24	897	8.0	11,045,039	22,139,683
1903	*Nil*	2,978	41	14	1,273	10.0	11,498,111	22,157,521
1904	*Nil*	2,337	103	22	1,435	1.7	11,868,245	21,987,651

Year.	Copper.			Copperas.	Gold.		Graphite.	Iron.	
	Ore.	Metal.	Sulphate.		Ore.	Bullion.		Ore.	Pig.
1895	7,435	865	246	160	104	$49,841	28,443	1,384,911	660,549
1896	6,823	1,001	265	170	416	46,386	35,972	1,448,615	693,188
1897	7,405	1,083	276	125	647	44,924	38,504	1,613,876	762,685
1898	6,791	1,041	209	360	448	47,515	33,062	1,733,649	837,767
1899	6,731	1,123	235	475	387	50,306	31,819	1,725,143	872,352
1900	5,825	881	234	474	227	47,183	33,663	1,894,458	879,132
1901	7,406	776	256	472	143	31,234	29,992	1,963,246	884,844
1902	8,455	914	248	271	74	4,652	29,527	1,742,498	991,827
1903	12,688	961	310	298	2,148	5,316	29,590	1,715,984	970,832
1904	16,201	889	808	414	12,653	47,183	28,620	1,719,219	988,364

Year.	Lead.			Manganese Ore.	Mineral Paint.	Petroleum.	Quicksilver.		Salt.
	Ore.	Pig.	Litharge.				Ore.	Metal.	
1895	12,919	8,085	2,034	4,352	3,164	188,634	86,683	535	278,875
1896	14,563	9,769	1,738	3,950	3,979	262,356	83,305	564	308,933
1897	14,145	9,680	1,626	6,012	3,653	275,204	88,238	532	331,084
1898	14,363	10,340	1,520	6,132	3,213	323,142	88,519	491	341,959
1899	12,820	9,736	1,526	5,411	2,055	309,590	92,323	536	342,059
1900	14,314	10,650	1,288	8,804	2,828	347,213	94,747	510	330,277
1901	16,688	10,161	1,317	7,796	1,701	404,662	97,360	525	333,238
1902	19,055	11,264	1,023	5,646	1,486	520,845	90,040	511	311,806
1903	22,196	12,162	923	6,179	1,691	672,508	83,321	523	359,015
1904	22,514	12,645	783	10,189	1,829		88,279	536	369,877

Year.	Silver.		Sulphuric Acid.	Sulphur Ore.	Tin.		Tungsten Ore.	Uranium.		Zinc.	
	Ore.	Bullion.			Ore.	Block.		Ore.	Salts.	Ore.	Spelter.
1895	18,113	40,081	7,431	830	24	60	35	31	4.5	25,862	6,456
1896	18,701	39,904	7,972	643	15	54	22	30	4.2	26,887	6,888
1897	20,628	40,026	8,515	530	16	48	31	44	4.4	27,463	6,236
1898	20,886	40,304	7,003	496	13	48	36	51	4.3	27,395	7,302
1899	21,554	39,564	7,814	555	54	41	50	49	7.6	37,100	7,192
1900	21,641	39,572	7,067	862	51	40	50	52	11.3	38,243	6,742
1901	21,363	40,205	7,073	4,911	42	49	45	48	13.0	36,072	7,558
1902	22,288	39,544	8,781	3,721	47	50	45	46	10.0	31,927	8,309
1903	21,958	39,812	9,105	4,475	57	34	49	45	6.0	29,544	8,949
1904	21,949	39,032	8,742	6,288	77	38	52	17	11.0	29,226	9,159

(*a*) From the ***Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.***

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*)

(In metric tons or dollars; 1 Crown=$0.203.)

Year.	Antimony.		Asphalt.	Asphaltic Rock.	Bismuth	Carbon Bi-Sulphide.	Coal.			
	Ore.	Regulus.					Bituminous (*e*).	Lignite (*e*).	Coke.	Briquettes.
1895	1,240	465	2,285	....	....	237	1,068,046	3,517,901	12,033	29,421
1896	1,361	500	2,740	....	....	352	1,132,625	3,761,728	25,550	31,179
1897	1,800	523	3,057	....	4.7	432	1,118,024	3,870,530	(*d*)	27,022
1898	2,201	855	3,125	....	3.1	771	1,239,498	4,516,581	(*d*)	31,781
1899	1,965	940	3,060	....	3.0	1,120	1,238,855	4,292,584	10,336	31,137
1900	2,373	846	2,700	....	2.0	1,250	1,447,047	5,128,277	12,973	69,353
1901	(*b*) 323	706	2,878	25,161	1.6	2,087	1,365,270	5,179,829	10,975	40,182
1902	(*b*) 748	683	2,774	24,873	0.9	2,320	1,162,785	5,132,053	8,204	88,069
1903	(*b*) 205	732	2,422	21,552	1.5	2,357	1,233,410	5,271,781	9,442	101,197
1904	1,080	1,007	2,221	17,660	0.9	2,512	1,155,320	5,519,349	(*d*)	103,481

Year.	Copper.	Copperas.	Gold.	Iron.			Lead.		Litharge.	Manganese Ore.
				Ore. (*e*).	Pig.	Cast.	Ore.	Pig.		
1895	286	521	$2,118,100	9,955,262	322,206	....	.....	2.277	615	3,525
1896	159	595	2,131,876	1,269,680	383,698	....	.....	1,911	465	2,101
1897	213	592	2,038,839	1,421,130	402,503	....	525	2,527	155	4,030
1898	153	745	1,839,474	1,666,837	448,621	20,784	771	2,305	188	8,087
1899	165	771	2,039,504	1,587,600	451,637	19,631	526	2,166	213	5,073
1900	181	700	2,173,079	1,666,363	432,817	22,738	612	2,030	201	5,746
1901	162	805	2,189,692	1,557,300	430,686	20,640	(*b*) 10	2,029	238	4,591
1902	89	909	2,260,135	1,562,238	416,835	18,569	(*b*) 20	2.244	219	7,237
1903	45	982	2,243,521	1,439,132	396,674	18,875	(*e*)3,698	2,057	257	5,311
1904	63	1,277	2,437,998	1,524,036	370,297	17,203	(*e*)3,922	2,104	(*d*)	11,527

Year.	Mineral Paints.	Petroleum (*e*).	Pyrites.	Quicksilver *Kg.*	Salt.	Silver-*Kg.*	Sulphur	Sulphuric Acid.	Zinc.	
									Ore. (*b*)	Spelter.
1895	371	2,083	69,195	1,129	169,395	20,432	102	4,223	(*d*)	..
1896	334	2,168	52,697	1,100	180,133	19,916	138	3,550	(*d*)	..
1897	460	2,229	44,454	700	171,711	26,790	112	3,397	30	..
1898	247	2,471	58,079	6,800	178,551	18,799	93	1,318	30	..
1899	394	2,125	79,519	27,000	182,593	20,991	116	1,463	1,197	..
1900	370	2,199	87,000	31,800	189,363	20,202	123	1,371	326	..
1901	305	3,296	93,907	33,003	184,083	23,636	137	1,464	693	14
1902	283	4,347	106,490	44,600	174,882	23,020	105	1,193	364	..
1903	263	3,010	96,619	43,700	183,328	19,281	135	1,543	46	26
1904	273	2,134	97,148	45,169	(*d*)	16,352	143	1,329	203	..

(*a*) From the *Annuaire Statistique Hongrois.* (*b*) Includes only that part of the crude output that was not smelted into a refined product. (*d*) Not reported. (*e*) Total production.

MINERAL AND METALLURGICAL PRODUCTION OF BOSNIA AND HERZEGOVINA. (a)

(In metric tons.)

Year	Chrome Ore.	Copper.		Iron.		Lignite	Manganese Ore.	Pyrites.	Quicksilver.	Salt.
		Ore.	Metal.	Ore.	Pig.					
1895....	707	(b)	105	(b)	2,569	195,422	8,145	....	(b)	12,758
1896....	443	(b)	296	(b)	10,120	222,724	6,821	....	(b)	13,720
1897 ...	396	3,847	135	37,095	15,606	229,643	5,344	....	(b)	13,919
1898....	458	3,760	156	57,935	15,263	270,752	5,320	3,670	4.0	14,496
1899....	200	3,980	180	67,030	13,730	303,000	5,270	....	3.3	15,030
1900....	100	3,008	141	133,454	38,960	394,516	7,939	1,700	6.7	15,791
1901....	505	3,696	199	122,569	39,296	445,007	6,346	4,570	9.3	16,865
1902....	270	3,657	166	133,348	43,992	424,753	5,760	5,170	7.2	17,348
1903....	147	1,073	191	114,059	39,833	467,962	4,538	6,589	8.1	18,459
1904....	279	640	115	127,297	47,678	483,617	1,114	10,421	8.1	18,021
1905....	186	670	39	122,540	43,074	540,237	4,129	19,045	10	(b)

(a) From *Oesterr. Zeits., für Berg-, Hütten- und Salinenwesen.* (b) Not reported.

MINERAL IMPORTS OF AUSTRIA-HUNGARY. (a)

(In metric tons or dollars; 5 Crowns=$1.)

Year.	Alum.	Aluminum and Alloys.	Aluminum, Sulphate and Chloride.	Ammoniacal Liquor.	Ammonium.		Antimony.	
					Chloride and Sulphate.	Hydrate.	Ore.	Regulus—*Kg.*
1895..........	338	48	1,278	877	305	103	15	2,100
1896..........	359	50	1,128	507	323	71	16	700
1897..........	346	67	1,351	565	339	128	8	600
1898..........	338	101	1,822	230	430	80	12	28,200
1899..........	332	121	1,299	73	358	46	10	30,400
1900..........	430	154	1,435	176	573	65	46	23,000
1901..........	413	153	1,882	293	620	44	27	1,500
1902..........	537	151	2,161	402	438	22	40	18,200
1903..........	508	150	2,670	580	361	23	42	87,200
1904..........	602	231	2,346	426	388	20	64	21,000

Year.	Arsenic. (b)	Asbestos.		Asphalt.		Barytes.	Borax.	
		Crude (c).	Manufactures	Crude Rock.	Mastic and Bitumen.		Crude and Boric Acid.	Refined.
1895.......	293	432	108	2,410	872	5,098	1,908	62
1896.....	309	185	165	4,715	1,621	5,377	1,363	76
1897.......	259	625	134	5,824	1,309	4,947	1,206	63
1898.......	287	609	138	5,973	1,117	5,012	784	185
1899.......	284	866	1347	7,301	1,546	5,443	2,212	130
1900.......	320	1,085	1238	8,301	1,564	5,945	3,056	93
1901.......	351	1,678	1032	5,702	1,106	6,336	1,687	233
1902.......	351	2,038	798	5,732	1,273	6,266	2,168	174
1903.......	371	3,395	1221	5,871	1,272	7,057	2,192	150
1904.......	384	2,517	1240	8,211	1,064	7,883	2,572	142

Year.	Brass, German Silver and Tombac.			Cement.	Chloride of Lime.	Chrome Ore.	Clay Products.		
	Crude, Old and Scrap.	Bars, Sheets and Wire.	Wares.				Kaolin and Feldspar.	Manufactures.	Potters' and Other Clays.
1895....	2,742	131	510	32,012	2,538	1,827	6,532	194,476	27,493
1896....	3,118	113	526	35,290	1,989	1,891	7,425	213,208	30,072
1897....	2,660	152	549	32,479	1,820	1,109	6,913	186,297	28,925
1898....	3,232	182	607	30,745	2,851	2,206	7,991	183,822	31,905
1899....	2,699	168	588	21,410	3,749	1,874	8,152	177,119	30,799
1900....	2,654	54	579	25,747	3,326	2,823	6,847	179,699	30,419
1901....	3,771	101	577	23,559	3,326	860	7,687	190,585	36,448
1902....	3,744	409	625	18,658	2,596	2,668	9,085	209,734	28,985
1903....	3,716	612	658	23,256	2,791	2,121	9,940	234,008	32,015
1904....	4,761	573	769	20,259	3,407	1,209	10,854	250,568	33,281

Year.	Coal.		Coke.	Copper.			Copper Sulphate.	Cop-peras.	Cryolite.
	Bituminous.	Lignitic.		Ore.	Bullion and Scrap.	Bars, Sheets, Wire, etc.			
1895.......	4,503,003	16,797	533,402	31	11,747	98	895	871	229
1896.......	5,174,321	19,981	491,028	1	13,666	126	2,084	575	265
1897	5,121,475	19,609	533,463	81	15,926	94	6,822	401	211
1898... ...	5,396,760	19,393	606,783	64	17,443	159	5,271	466	275
1899..	5,296,700	20,879	564,005	*Nil*	16,185	156	2,345	409	343
1900.......	6,242,939	67,740	620,776	16	18,970	121	3,516	343	342
1901.......	5,827,332	22,253	612,209	112	17,504	83	2,822	269	428
1902.	5,766,377	29,601	547,406	100	18,498	149	2,839	274	447
1903..	5,907,660	30,007	519,281	209	18,701	89	3,526	155	521
1904	6,190,030	30,001	548,272	1,107	22,532	89	4,508	238	313

Year.	Fertilizers, Mineral.	Fluorspar.	Glass (all kinds).	Gold. (*d*)			Graph-ite.	Gypsum.	
				Bullion.	Coin.	Old and Dross.		Crude.	Burned.
1895....	....	3,528	. . .	$3,470,945	$17,384,964	$76,290	640	850	10,916
1896....	......	3,821	....	8,674,371	16,956,256	17,260	697	821	11,736
1897....	......	4,201	....	22,374,069	18,164,128	23,120	948	980	12,101
1898....	......	4,169	....	323,636	8,853,354	11,520	1,109	991	13,300
1899....	80,658	4,959	4,422	432,187	7,662,641	6,902	815	1,336	13,441
1900....	71,966	5,649	3,954	1,111,831	7,230,251	8,932	302	1,348	15,462
1901....	93,430	6,575	4,594	13,865,103	20,353,592	14,819	318	1,405	15,830
1902....	99,502	6,851	4,608	14,509,019	15,695,960	12,789	221	1,588	16,430
1903....	128,320	6,431	3,882	9,825,200	9,817,283	14,819	405	1,969	18,655
1904....	175,681	7,975	4,229	12,703,740	7,794,794	22,127	423	2,384	19,387

Year.	Hydro-chloric Acid.	Iron.				Lead.					
		Ore.	Pig and Old.	Manufac-tures.	Iron and Steel Bars, Sheets, Wire, etc.	Alloys, Crude.	Pig.	Lith-arge.	Ore.	Red and Yellow.	White.
1895...	467	117,600	175,400	$3,990,400	30,909	8,974	208	355	416	371	187
1896...	529	107,018	148,217	4,258,400	27,809	7,221	218	233	540	432	156
1897...	721	134,778	164,433	4,582,400	18,625	5,887	148	224	441	543	111
1898...	766	178,507	173,919	4,627,200	26,421	9,746	153	280	459	555	115
1899...	350	212,412	126,371	4,395,356	12,340	8,836	235	224	465	466	80
1900...	577	233,156	95,530	4,533,599	10,313	7,916	175	141	501	354	106
1901...	576	218,476	90,287	4,443,670	10,902	10,722	311	189	1,270	433	135
1902...	588	197,525	43,314	4,304,818	11,584	8,706	348	149	1,879	428	221
1903...	603	217,979	47,354	4,508,224	11,025	9,190	409	141	1,355	423	173
1904...	459	182,515	35,091	4,976,342	9,402	7,917	349	146	1,436	372	138

Year.	Magnesium Chloride.	Manganese Ore.	Millstones.	Mineral Paints.	Nickel.		Nickel and Cobalt Ores.	Nitric Acid.	Peat and Peat Coke.
					Old and Crude.	Sheet, Wire etc.			
1895......	1,353	2,772	1,229	4,244	168	5	1,020	16	1,993
1896......	1,333	7,371	1,205	4,362	161	5	719	21	2,002
1897......	1,530	8,018	1,275	4,553	157	7	55	23	2,189
1898.	2.096	5,396	1,429	4,979	137	9	510	22	1,511
1899......	2,043	5,855	1,458	5,106	119	11	198	39	2,075
1900......	2,100	7,016	1,672	4,958	258	8	406	36	2,664
1901......	2,529	6,367	1,595	5,109	277	10	788	22	2,896
1902.. ..	2,621	15,595	1,410	4,831	265	8	225	90	3,234
1903......	3,118	38,529	1,395	4,733	268	9	385	7	3,097
1904......	2,997	35,357	1,282	5,563	402	6	656	24	2,676

Year.	Petroleum Products.			Phosphorus and Phosphoric Acid.	Potassium Salts.			Pyrites.
	Crude Oil.	Refined Oil.	Paraffin.		Carbonate.	Chloride.	Chromate.	
1895.........	120,479	16,876	.....	226	285	2,679	29	54,610
1896.........	69,013	17,943	.....	224	987	2,475	34	50,691
1897.........	70,573	21,249	.....	209	333	2,206	34	49,462
1898.........	58,580	22,299	.....	209	300	2,258	3	52,282
1899.........	75,885	21,823	6,968	221	526	3,264	1	54,844
1900.........	20,813	22,963	5,080	204	1,029	3,633	11	60,317
1901.........	22,545	18,067	5,294	222	1,442	4,356	21	54,202
1902.........	24,830	15,864	4,238	225	485	3,377	11	60,235
1903.........	19,710	19,382	2,598	237	197	3,727	0	73,835
1904.........	20,110	22,715	1,470	193	360	3,557	3	65,397

Year.	Quicksilver *Kg.*	Salt.	Silica, Quartz and Sand.	Silver.			Slag and Slag Wool.
				Bullion, *Kg.*	Old and Dross, *Kg.*	Specie.	
1895.............	4,200	40,396	58,494	49,370	60	$90,353	981
1896.............	1,300	53,680	59,150	138,420	550	137,244	240
1897.............	1,000	46,057	61,532	99,900	1,000	75,944	4,717
1898.............	6,300	41,870	70,296	15,400	3,000	103,424	9,655
1899.............	2,600	37,883	71,279	28,900	600	112,056	5,665
1900.............	1,300	39,822	77,930	29,300	2,700	199,955	4,679
1901.............	2,600	39,625	83,401	41,800	1,700	207,669	3,068
1902.............	1,300	46,128	92.617	177,900	1,500	237,104	4,176
1903.............	1,600	48,793	94,492	150,400	5,700	250,299	3,850
1904.............	2,500	94,103	97,364	36,700	1,700	420,413	4,716

Year.	Slate and Other Roofing.	Sodium Salts.					
		Bi-Sulphate.	Carbonate.	Carbonate. (Calcined).	Hydrate.	Nitrate.	Sulphate.
1895.......	15,667	137	40	551	1,163	43,059	6,617
1896.......	15,932	144	57	1,332	835	33,086	4,678
1897.......	16,758	91	45	2,787	1,450	39,600	2,879
1898.......	16,025	89	53	2,408	1,498	41,773	4,476
1899	15,562	85	62	1,123	1,669	47,301	5,394
1900.......	13,047	73	104	1,141	1,836	54,559	5,110
1901.......	11,555	98	77	911	1,293	63,283	4,452
1902.......	14,378	17	97	312	1,071	39,958	5,997
1903.......	11,531	13	110	327	970	54,896	6,116
1904.......	9,170	103	103	1,109	673	54,887	5,409

Year.	Stone.				Sulphur.	Sulphuric Acid.
	Lithographic.	Marble.	Paving.	Not Elsewhere Specified.		
1895..	684	1,886	5,459	79,869	14,709	1,566
1896	647	2,347	8,476	87,796	15,221	3,522
1897.......................	524	2,333	16,961	118,848	21,406	5,877
1898	786	2,769	11,234	99,193	20,655	9,724
1899.......................	611	2,850	27,067	82,878	23,504	10,245
1900.	640	3,133	5,781	81,347	27,795	10,643
1901.......................	616	2,908	11,299	77,695	25,300	11,712
1902	363	3,184	10,203	88,214	23,878	12,474
1903.......................	668	3,132	14,266	74,960	22,625	16,148
1904.......................	679	4,016	12,177	89,791	30,505	19,878

Year.	Tin.		Whetstones.	Zinc.			
	Ingot, Crude, Old, etc.	Salts		Calamine and Other Ores.	Spelter.	Bars, Sheets Wire, etc.	White.
1895	3,038	46.0	3,559	7,691	17,156	611	510
1896............	3,344	27.7	3,851	9,022	17,539	552	590
1897............	3,467	22.3	4,151	7,863	16,599	356	577
1898..	3,769	20.9	3,490	14,112	17,471	453	697
1899............	3,005	30.2	3,717	12,730	15,225	481	750
1900............	3,439	23.9	3,643	14,181	17,844	667	875
1901.	3,671	24.9	3,445	18,403	16,921	579	718
1902............	3,638	31.7	3,599	20,723	17,034	651	636
1903............	3,564	31.3	3,774	22,344	17,973	746	698
1904	3,528	54.8	4,272	24,039	20,787	731	840

(*a*) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (*b*) Includes arsenious acid and sulphide. (*c*) Includes burned asbestos. (*d*) The values of gold are figured at the rate of one crown=$0.203.

MINERAL EXPORTS OF AUSTRIA-HUNGARY. (a)

(In metric tons or dollars; 5 crowns=$1.)

Year.	Alum.	Aluminum, Sulphate and Chloride.	Ammonium.		Ammoniacal Liquor.	Antimony.		Arsenic, Arsenious Acid, and Orpiment.
			Chloride and Sulphate.	Hydrate		Ore.	Regulus.	
1895	60	231	876	92	413	193	369	36
1896	47	267	2,524	70	604	218	441	26
1897	70	210	4,188	39	592	289	359	16
1898	83	253	4,886	23	724	266	679	29
1899	54	233	7,576	42	734	562	240	47
1900	44	164	7,004	82	942	247	276	65
1901	55	211	8,824	113	705	179	385	80
1902	102	135	11,777	93	717	174	290	89
1903	77	14	11,478	155	814	128	249	63
1904	38	2	12,386	130	654	200	673	72

Year.	Asbestos.		Asphalt.		Barium.		Brass, German Silver and Tombac		Chloride of Lime.
	Crude.	Manufactured.	Rock and Earth.	Mastic and Bitumen.	Sulphate. (b)	Chloride.	Crude, Old and Scrap.	Bars, Sheets, Wire, etc.	
1895	122	10	145	1,183	..	.....	1,115	459	267
1896	48	10	134	1,692	..	.....	1,469	399	114
1897	56	19	102	2,593	..	.....	1,699	680	111
1898	150	28	183	2,126	..	.....	1,879	839	113
1899	71	60	1,143	2,619	65	.....	1,824	910	203
1900	47	168	1,218	2,177	23	.....	1,811	1,256	192
1901	36	165	198	1,909	55	4,098	2,006	1,054	738
1902	65	275	520	301	64	4,552	2,298	1,098	426
1903	89	495	921	483	52	5,091	2,576	853	674
1904	370	1,582	403	728	74	4,233	2,715	1,031	254

Year.	Cement.	Chrome Ore.	Clay Products.			Coal.		Coke.
			Manufactures.	Kaolin and Feldspar.	Potters' and other Clays.	Bituminous.	Lignitic.	
1895	12,804	385	51,767	56,203	37,667	640,963	7,143,234	119,051
1896	16,721	142	67,270	67,381	41,276	658,368	7,562,721	116,608
1897	19,786	153	55,895	68,609	46,968	701,919	8,108,975	145,056
1898	23,989	121	66,005	74,003	52,851	824,730	8,351,955	194,289
1899	38,193	53	72,104	78,537	61,898	879,337	8,662,788	252,971
1900	46,761	22	75,672	103,178	66,869	815,097	7,864,410	262,793
1901	44,723	62	62,916	97,037	57,571	748,802	8,076,575	303,651
1902	39,920	51	56,244	100,546	52,664	691,680	7,888,218	234,911
1903	40,239	100	65,004	110,181	50,371	754,957	8,027,347	280,395
1904	43,110	36	73,379	127,984	57,143	815,570	7,588,555	353,695

Year.	Fertilizers.	Fluorspar.	Glass. (All kinds.)	Copper.			Copper Sulphate.	Copperas.	Cryolite.
				Ore.	Crude and Old.	Bars, Sheets Plates, etc.			
1895	.....	44	......	17	151	354	162	301	11
1896	.....	40	......	12	228	189	47	392	2
1897	.....	27	......	0.1	159	180	14	648	10
1898	.....	22	......	12	173	266	29	539	23
1899	5,468	309	55,649	74	534	298	67	808	101
1900	5,385	45	57,281	801	471	200	57	748	237
1901	5,628	6	55,409	1,042	435	334	23	548	231
1902	5,554	42	56,125	1,018	436	381	44	857	363
1903	10,850	12	61,210	1,308	1,226	451	45	898	521
1904	10,147	36	63,643	574	747	577	50	1,170	574

Year.	Gold.				Graphite.	Gypsum.		Hydrochloric Acid.
	Ore.	Bullion.(e)	Old and Dross.(e)	Specie.(e)		Crude.	Calcined.	
1895	1	$203,352	$236,487	$ 8,885,815	11,923	1,496	1,439	1,460
1896	45	253,194	235,935	13,555,706	13,091	899	1,376	1,246
1897	37	158,827	357,798	18,598,931	14,229	662	1,804	1,439
1898	13	17,943	377,336	23,779,858	17,109	718	2,163	1,614
1899	67	17,864	313,432	12,711,454	19,451	634	1,539	1,495
1900	1.2	120,988	316,883	11,582,571	18,995	502	1,723	1,659
1901	0.2	42,427	352,408	6,880,888	14,900	461	1,206	1,632
1902	..	22,939	359,107	13,485,087	16,771	550	1,041	791
1903	3.7	10,150	368,445	11,052,944	17,302	342	1,510	3,530
1904	64.1	5,278	370,272	9,649,605	17,430	392	1,510	3,722

Year.	Iron. Ore.	Iron. Pig and Old.	Iron. Manufactures.	Iron. Iron and Steel. Bars, Sheets, Wire, etc.	Lime. Hydraulic and Caustic.	Magnesium. Chloride and Glauber Salts.	Magnesite (Calcined.)
1895	165,402	9,786	18,698	9,993	34,698	661	(c)
1896	214,390	11,712	17,674	12,428	76,895	2,291	(c)
1897	247,856	12,084	21,064	17,387	83,110	6,910	(c)
1898	302,317	15,803	22,724	23,231	89,067	7,248	(c)
1899	326,951	27,738	30,822	50,197	85,570	5,721	(c)
1900	263,421	53,426	40,344	65,019	86,273	7,321	(c)
1901	229,624	26,304	46,508	28,841	82,399	7,960	40,236
1902	241,806	42,592	30,137	45,517	81,634	5,333	53,467
1903	252,520	60,237	40,807	63,031	95,644	2,360	69,058
1904	295.017	66,442	58,648	64,698	101.753	2,151	53,781

Year.	Lead. Ore.	Lead. Ash.	Lead. Litharge.	Lead. Metal and Alloys.	Lead. Red and Yellow	Lead. White.	Manganese Ore.	Millstones.	Mineral Paints.	Nickel Bars, Sheets, Wire, etc.
1895	3,758	118	782	208	24	233	425	1,977	2,244	54
1896	3,076	113	597	272	33	171	701	1,831	1,700	273
1897	2,438	114	355	241	24	47	622	1,773	1,621	170
1898	2,253	100	188	545	45	55	1,961	2,109	2,153	76
1899	2,502	99	188	258	45	41	1,127	1,904	2,061	38
1900	2,628	66	242	393	21	34	463	1,871	1,906	119
1901	4,143	112	179	68	32	23	398	1,971	1,947	97
1902	5,478	154	124	109	25	37	411	1,886	2,136	75
1903	8,961	147	145	152	19	25	724	2,311	1,873	124
1904	7,575	144	167	464	54	52	1,234	2,276	1,840	36

Year.	Nickel and Cobalt Ores.	Nitric Acid.	Oxide. (Potash)	Ozokerite.	Peat and Peat Coke.	Petroleum Products. Petroleum. (d)	Petroleum Products. Benzine.	Petroleum Products. Paraffin.	Potassium Chloride.	Pyrites.
1895	139	418	5,665	5,054	3,753	5,317	....	....	1,074	383
1896	113	360	4,164	5,722	2,701	24,921	....	....	1,026	341
1897	117	310	5,997	5,153	1,655	14,682	....	....	1,005	255
1898	121	294	7,252	4,462	3,400	4,138	....	....	994	3,039
1899	75	420	10,113	5,441	4,010	11,756	20,646	10	974	5,201
1900	114	519	7,792	5,162	5,607	33,032	18,361	26	879	17,162
1901	120	632	4,234	2,717	4,558	19,804	17,021	14	909	16,491
1902	34	769	3,229	2,285	4,927	40,683	13,884	24	772	9,547
1903	12	908	3,409	2,258	3,638	74,454	14,000	1,153	802	10,857
1904	26	858	4,604	2,093	3,980	122,419	13,706	5,992	445	9,891

Year.	Stone. Lithographic.	Stone. Limestone.	Stone. Marble.	Stone. Paving.	Stone. Not Elsewhere Specified.	Sulphur.	Sulphuric Acid.
1895	4	33,479	2,737	29,392	241,040	989	6,466
1896	6	23,249	3,595	30,577	216,666	1,231	6,212
1897	22	11,373	2,994	38,256	223,321	947	7,903
1898	8	25,117	2,954	54,953	252,229	923	9,880
1899	11	11,053	4,096	67,212	239,819	885	12,422
1900	7	13,878	3,811	58,121	197,932	1,285	12,693
1901	2	23,909	4,394	51,613	200,461	1,225	10,373
1902	4	29,047	4,367	33.689	230,301	1,136	9,451
1903	2	26,951	5,628	39,388	229,060	1,123	8,369
1904	6	20,331	6,220	52,720	226,874	988	9,101

Year.	Tin. Ingot and Old.	Tin. Bars, Plates, Sheets, etc.	Tin. Ash.	Whetstones.	Zinc. Ores.	Zinc. Crude and Old.	Zinc. Sheets, etc.	Zinc. White.	Zinc. Ash.
1895	53	90	248	2,169	7,491	504	1,158	1,688	179
1896	130	78	281	2,035	9,453	1,256	1,139	1,825	277
1897	87	75	306	2,323	12,914	770	993	1,673	197
1898	96	72	324	2,316	14,065	1,184	757	1,240	298
1899	167	77	273	2,215	20,461	1,614	1,313	1,096	73
1900	153	102	208	2,270	20,379	1,088	502	1,719	149
1901	162	109	257	2,196	23,150	1,374	577	2,720	167
1902	193	128	188	2,651	24,519	2,002	1,076	3,113	237
1903	292	111	158	2,392	15,108	4,420	699	3,446	267
1904	126	102	123	1,953	17,314	4.606	481	3,666	158

(*a*) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (*b*) Includes artificial barium sulphate. (*c*) Previous to 1901, magnesite was included with other minerals not elsewhere specified. (*d*) From 1895 to 1898 inclusive, includes crude and refined petroleum; from 1899 to 1904 inclusive, lubricating oil is also included. (*e*) Where gold or silver values are reported 1 crown=$0.203.

BELGIUM.

The production and foreign commerce of mining and metallurgical products in Belgium are reported as follows:

MINERAL, METALLURGICAL AND QUARRY PRODUCTION OF BELGIUM. (*a*)
(In metric tons except where otherwise noted.)

Year.	Barytes.	Chalk, Marl, *Cubic Meters.*	Clay.	Coal.		Coke.	Flint. *Cubic Meters.*		Iron Ore.
				Bituminous.	Briquettes.		For Earthenware.	For Ballast. (*c*)	
1895....	32,750	100,160	195,485	20,457,604	1,217,795	1,794,109	24,870	202,590	312,637
1896....	25,000	191,100	83,020	21,252,370	1,213,760	2,004,430	23,450	244,050	307,031
1897....	23,000	204,600	270,715	21,492,446	1,245,114	2,207,840	23,050	235,495	240,774
1898....	21,700	287,805	287,805	22,088,335	1,351,884	2,161,162	22,150	360,960	217,370
1899....	25,900	351,800	291,125	22,072,068	1,276,050	2,304,607	25,185	258,835	201,445
1900....	38,800	377,550	313,205	23,462,817	1,395,910	2,434,678	25,700	263,850	247,890
1901....	22,800	449,000	298,340	22,213,410	1,587,800	1,847,780	17,700	7,860	218,780
1902....	33,000	390,700	299,820	22,877,470	1,616,520	2,048,070	17,430	7,705	166,480
1903....	21,000	501,920	292,855	23,796,680	1,686,415	2,203,020	16,250	8,935	184,400
1904....	60,000	450,400	347,135	22,761,430	1,735,480	2,211,820	18,070	12,500	206,730

Year.	Iron, Crude.					Iron, Manufactures of.			
	Forge Pig.	Foundry Pig.	Bessemer Pig.	Basic Pig.	Total Pig.	Merchant Bars.	Sheet and Plate.	Wrought	Other Mfres.
1895........	329,750	85,450	161,606	252,428	829,234	76,101	109,209	741	259,323
1896........	362,451	84,275	193,518	307,779	959,414	81,394	112,597	851	298,163
1897........	426,332	78,410	183,701	333,958	1,035,037	108,608	100,252	872	263,644
1898........	308,875	93,645	173,085	397,891	979,755	123,993	91,686	993	267,521
1899........	317,029	84,165	169,664	453,718	1,024,576	93,601	97,604	662	283,331
1900........	305,344	88,335	176,557	447,271	1,018,561	61,458	73,572	1,411	284,591
1901........	178,250	86,170	166,820	332,940	764,180	249,380	65,760	550	64,900
1902........	104,540	254,710	199,170	510,630	1,069,050	260,290	62,740	450	58,150
1903........	91,600	256,890	229,160	638,430	1,216,080	274,520	56,550	390	60,920
1904........	99,350	224,410	217,390	742,040	1,287,597	246,240	41,000	370	67,580

Year.	Steel.					Lead.		Manganese Ore.
	Ingots, Blooms and Billets.	Rails.	Tires.	Wrought.	Plates.	Ore.	Pig.	
1895.....................	454,619	122,357	7,359	4,551	42,444	220	15,573	22,478
1896.....................	598,947	147,183	10,497	6,702	64,653	70	17,222	23,265
1897.....................	616,541	136,911	10,870	23,104	64,366	108	17,023	28,372
1898.....................	653,523	117,751	10,953	17,902	87,219	133	19,330	16,440
1899.....................	731,249	123,119	11,212	32,180	68,051	137	15,727	12,120
1900.....................	655,199	134,428	11,934	25,985	55,307	230	16,365	10,820
1901.....................	529,840	132,260	12,380	3,310	84,730	220	18,760	8,510
1902.....................	786,980	(*e*)268,220	12,790	2,910	11,530	164	73,357	14,440
1903.....................	988,160	(*e*)351,540	17,810	2,920	67,160	90	68,700	6,100
1904.....................	1,065,870	(*e*)266,900	23,540	4,300	86,790	91	23,470	485

Year.	Mineral Paints. Ochers. *Cubic Meters.*	Phosphate of lime. *Cubic Meters.*	Pyrites.	Sand. *Cubic Meters*	Slate. Pieces.	Stone. *Cubic Meters.* Silver. *Kg.*	Dolomite.	Flagstones. *Sq. Meters.*	Freestone.
1895..	500	506,730	3,510	381,170	33,652,000	31,543	(*b*)	95,137	137,353
1896..	700	297,470	2,560	418,720	35,980,000	28,509	21,500	131,400	152,420
1897..	350	350,056	1,828	559,141	41,422,000	30,073	52,720	107,572	181,746
1898..	290	156,920	147	287,805	42,311,000	116,035	37,100	170,672	215,417
1899..	300	190,090	283	627,770	44,167,000	134,854	56,400	144,330	139,294
1900..	300	215,670	400	653,780	43,941,000	146,548	45,000	153,217	157,294
1901..	(*d*)2,100	(*d*) 222,520	560	626,020	39,030,000	169,450	31,500	106,470	167,310
1902..	(*d*) 200	(*d*) 135,850	710	722,775	37,120,000	212,249	39,140	101,945	185,319
1903..	(*d*) 200	(*d*) 184,120	720	724,495	38,953,000	232,740	43,600	117,165	245,184
1904..	(*d*) 450	(*d*) 202,480	1,075	802,115	41,240,000	252,920	48,600	71,630	216,717

Year.	Stone. *Cubic Meters.* (*Continued*). Limestone.	Limestone for Flux.	Marble.	Paving Stones. *Pieces.*	Whetstones and Hones. *Pieces.*	Zinc. Ore. (Blende)	Ore. (Calamine)	Spelter.	Sheets.
1895..........	2,488,840	163,800	12,790	92,378,800	70,000	8,080	4,150	107,664	34,081
1896..........	2,646,305	164,900	16,315	102,295,950	45,850	7,070	4,560	113,361	36,238
1897..........	3,010,877	225,300	17,797	95,542,700	43,150	6,804	4,150	116,067	37,011
1898..........	2,968,997	212,685	16,610	108,025,000	89,150	7,350	4,125	119,671	35,587
1899..........	3,238,875	195,505	17,740	114,103,900	82,100	5,736	3,730	122,843	34,289
1900..........	3,228,205	229,250	15,990	107,294,600	105,000	5,715	3,000	119,317	38,825
1901..........	3,751,880	193,370	15,390	110,920,000	160,150	4,445	2,200	127,170	37,380
1902..........	1,626,670	226,220	15,490	110,103,000	122,300	3,568	284	124,780	37,070
1903..........	1,580,330	210,250	16,735	111,318,000	134,620	3,565	65	131,740	42,280
1904..........	1,645,655	213,320	17,740	117,412,000	135,700	3,698	4	137,323	41,490

(*a*) From *Statistique des Industries Extractives et Metallurgiques et des Appareils à vapeur en Belgique.* (*c*) Includes some gravel. (*d*) Metric tons. (*e*) Includes beams.

MINERAL IMPORTS OF BELGIUM. (*a*)

(In metric tons or dollars; 5f.=$1.)

Year.	Ashes.	Cement.	Clay Products. Terra Cotta.	Common Pottery.	Coal.	Coal Briquettes.	Coke.
1895.............	6,173	23,198	(*b*) 40,531	2,344	1,530,364	3,452	362,834
1896.............	6,747	30,565	85,486	2,065	1,693,376	1,561	260,273
1897.............	10,870	17,681	86,493	2,115	2,017,344	632	269,606
1898.............	8,199	34,039	92,149	2,007	2,202,517	1,756	180,590
1899.............	15,818	18,649	99,156	2,856	2,844,274	10,722	296,508
1900.............	15,428	12,773	90,852	4,281	3,288,513	21,814	289,673
1901.............	14,802	13,558	81,359	2,223	2,930,874	17,160	154,247
1902.............	16,708	13,269	91,795	2,132	3,232,510	33,235	230,612
1903.............	37,178	19,698	107,457	2,095	3,554,807	43,835	308,877
1904.............	56,052	37,592	111,394	6,401	3,701,240	45,600	338,127

Year.	Copper and Nickel. Crude.	Hammered, Drawn or Rolled.	Wrought	Diamonds Crude and Uncut.	Fertilizers (All).	Glass and Glassware. Common (Bottles, Broken Glass, Etc.)	Gold (Including Platinum). All Other Kinds.	Ore. *Kg.*	Unwr'ght. *Kg.*	Specie.	Jewelry.
1895...	10,480	926	$168,720	....	(*c*) 43,017	6,078	$441,558	65	2.714	$ 616,280	$ 523,491
1896....	15,506	1,109	188,931	....	(*c*) 25,946	6,980	541,225	93	4,923	599,540	757,507
1897....	14,821	1,418	193,242	....	(*c*) 5,162	4,699	664,176	......	3,824	1,726,700	701,535
1898....	14,947	1,821	205,705	....	(*c*) 10,657	4,247	635,021	8,390	1,282	372,000	840,593
1899....	8,327	2,174	226,853	....	(*c*) 15,072	3,757	651,208	51	1,136	744,000	965,170
1900....	13,768	2,087	231,800	$8,051,200	163,229	5,671	663,400	1,250	1,728	459,420	921,600
1901....	11,381	1,780	272,600	8,463,200	149,984	5,446	623,200	......	......	....	1,332,800
1902....	14,197	1,998	251,400	8,537,600	158,924	6,129	539,600	....	......	....	1,221,800
1903....	13,602	2,035	313,400	15,557,800	188,389	5,209	557,200	....	......	....	1,190,000
1904....	13,422	2,267	490,400	16,059,400	157,402	5,267	614,600	8,516	2,385	256,680	1,009,843

Year.	Iron.				Steel.		Lead.	
	Ore.	Pig and Scrap.	Manufactures.	Tin Plate.	Ingots, Blooms, and Billets.	Manufactures.	Pig.	Manufactures.
1895	1,857,624	247,854	22,548	2,046	18,405	18,718	45,594	$ 21,097
1896	2,069,676	378,191	28,930	3,203	28,435	16,199	35,221	17,231
1897	2,544,377	354,178	32,390	3,875	25,370	27,016	43,840	91,580
1898	2,252,553	370,117	40,388	3,848	25,142	25,774	54,867	50,728
1899	2,621,152	423,968	41,894	3,900	11,666	34,622	60,649	191,508
1900	2,528,615	371,726	40,716	5,036	19,705	43,875	58,141	143,000
1901	1,768,441	222,230	58,020	(d)4,705	68,228	51,427	54,720	123,200
1902	2,550,347	348,337	43,625	(d)6,608	103,286	43,039	71,085	164,800
1903	3,054,808	387,884	52,305	(d)9,776	144,370	56,139	63,386	126,600
1904	3,359,430	388,732	42,662	(d)7,892	182,336	49,464	63,813	234,400

Year.	Lime.	Petroleum.		Resins and Bitumens, Not Specified.	Salt.	
		Crude.	Refined.		Crude.	Refined.
1895	9,083	1,824	159,980	226,133	81,188	40,625
1896	11,522	95	158,979	216,278	92,408	38,785
1897	13,184	988	149,501	237,570	96,805	39,193
1898	12,674	382	161,281	269,914	92,300	50,136
1899	12,811	2,479	166,404	264,718	81,324	50,647
1900	11,448	1,751	158,064	253,788	97,812	59,375
1901	11,288	305	160,327	244,866	93,043	48,974
1902	13,820	247	183,592	305,597	102,110	50,704
1903	13,525	237	210,905	285,305	108,886	52,272
1904	16,896	1,893	192,805	205,136	110,516	54,615

Year.	Silver.				Sodium Salts.		
	Ore.	Bullion–*Kg.*	Specie.	Jewelry.	Carbonate.	Nitrate.	Sulphate and Sulphite.
1895	1,297	7,936	$ 2,793,680	343,840	(e)	184,306	(e)
1896	1,477	8,980	6,461,840	415,967	(e)	194,202	(e)
1897	2,533	467,851	2,083,040	460,244	(e)	181,676	(e)
1898	461	299,369	7,655,200	449,244	(e)	152,164	(e)
1899	2,523	105,723	14,272,080	520,070	(e)	249,756	(e)
1900	922	11,366	7,324,560	513,800	20,603	153,318	49,020
1901	....	....	....	550,000	19,465	167,489	47,558
1902	....	....	....	546,200	13,584	135,937	60,721
1903	....	....	....	597,800	19,459	149,942	54,926
1904	....	....	....	635,800	28,006	157,005	44,122

Year.	Stone.						Tin.		Zinc.	
	Roofing Slate. *1000 Pieces.*	Building Stone, Including Marble and Alabaster.	Cut, Polished, Etc.	Paving.	All Other Kinds.	Sulphur.	Block.	Manufactures.	Spelter.	Manufactures.
1895	37,720	31,156	$ 53,450	4,249	83,569	18,438	3,216	$3,501	8,551	$13,095
1896	38,209	40,511	61,769	6,163	81,360	14,399	4,617	2,845	20,182	11,230
1897	38,754	47,929	111,313	13,197	182,950	13,261	1,609	2,634	16,320	10,661
1898	38,216	45,544	63,665	8,926	239,281	13,322	1,208	2,545	17,441	11,575
1899	35,888	49,498	83,372	7,835	216,231	8,449	1,113	4,373	11,058	11,436
1900	34,331	60,057	145,800	3,761	135,245	17,516	1,653	5,540	11,478	12,000
1901	35,516	65,509	120,800	5,367	81,006	14,775	1,841	4,000	13,896	11,200
1902	36,950	69,745	109,000	7,124	72,384	12,367	1,416	4,800	17,830	14,600
1903	34,068	83,165	153,800	5,371	103,239	21,637	2,677	4,000	20,586	15,000
1904	34,139	86,013	143,800	4,545	115,275	24,788	3,416	4,200	17,424	13,800

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays Etrangers, Brussels.* (*b*) Pieces. (*c*) Guano only is reported. (*d*) Includes wrought tin-plate. (*e*) Included under Nitrate.

MINERAL EXPORTS OF BELGIUM. (*a*)
(In metric tons or dollars; 5 fr.=$1.)

Year.	Ashes.	Cement.	Clay Products.		Coal.	Coal Briquettes.	Coke.	Copper and Nickel.		
			Terra-cotta.	Common Pottery.				Crude.	Hammered Drawn or Rolled.	Wrought.
1895..	1,030	273,807	(*b*)127,413	3,463	4,661,477	459,702	870,983	5,205	2,558	$129,892
1896..	1,084	277,615	302,526	2,628	4,649,799	459,974	863,067	11,700	2,073	168,524
1897..	2,675	322,024	294,815	3,197	4,448,544	615,074	909,486	9,994	1,996	198,665
1898..	615	419,132	247,970	3,186	4,579,955	666,265	878,435	8,511	1,770	161,726
1899..	400	445,602	328,733	3,216	4,568,938	525,625	1,008,470	4,665	2,111	261,060
1900..	2,148	408,284	289,759	5,915	5,260,993	604,864	1,073,315	8,411	2,097	276,600
1901..	3,401	492,882	220,326	6,661	4,820,300	714,455	829,421	6,309	1,988	198,200
1902..	4,438	542,547	256,555	3,815	5,078,278	671,700	824,256	7,320	1,656	235,400
1903..	3,202	599,092	342,149	5,826	4,923,368	623,691	841,142	7,638	1,702	256,400
1904..	864	588,295	329,616	3,450	5,067,037	539.364	879,883	7,255	2,243	182,200

Year.	Diamonds. Crude and Uncut.	Fertilizers. (All Kinds.)	Glass and Glassware.			Gold (Including Platinum).		
			Common (Bottles, Broken Glass, etc.)	Plate.	All Other Kinds.	Unwrought, *Kg*.	Specie.	Jewelry.
1895......	....	(*c*) 18,968	4,181	$2,768,243	156,304	893	$3,547,640	$ 77,426
1896......	....	(*c*) 14,633	3,647	3,468,668	178,611	3,713	2,666,620	75,570
1897......	....	(*c*) 14,044	3,546	3,761,219	174,232	2,547	605,120	126,385
1898......	....	(*c*) 21,626	2,747	3,007,865	25,226	1,231	578,120	118,897
1899......	....	(*c*) 18,213	3,850	4,942,298	196,842	504	998,200	118,916
1900......	$ 8,601,000	258,366	3,993	4,413,000	159,557	549	613,180	161,200
1901......	8,814,200	299,709	5,486	5,030,400	154,610	......	....	225,800
1902......	8,855,400	359,607	6,338	4,799,800	198,647	......	....	193,800
1903......	16,708,200	497.355	5,156	4,660,800	197,246	......	....	159,400
1904......	17,028,400	555,956	6,781	4,707,200	146,637	671	726,020	137,400

Year.	Iron.				Steel.		Lead.		Lime.
	Ore.	Pig and Scrap.(*d*)	Manufactures.	Tin Plate.	Ingots, Blooms and Billets.	Manufactures.	Pig.	Manufactures.	
1895........	325,809	47,244	$ 9,680,461	1,750	1,315	197,766	39,996	$18,192	436,635
1896........	389,235	63,906	12,569,898	3,952	1,145	218,179	31,366	36,821	477,213
1897........	410,817	60,678	13,599,204	1,191	1,201	220,739	35,988	33,286	520,588
1898........	384,047	40,522	9,139,480	973	1,018	208,901	40,302	16,450	546,199
1899........	318,415	76,047	15,834,350	1,436	1,259	187,009	41,618	37,022	537,357
1900........	420,180	79,172	15,543,000	940	975	174,998	46,566	9,000	617,666
1901........	327,499	70,027	14,083,400	(*e*) 642	290	199,387	47,971	6,600	579,123
1902........	368,560	96,302	15,192,400	(*e*)1,023	1,463	310,802	58,495	9,400	623,617
1903........	400,972	101,549	16,059,000	(*e*)3,577	3,047	403,588	57,765	8,200	659,125
1904........	441,059	85,489	17,302,800	(*e*)2,378	5,250	331,755	59,344	13,400	706,351

Year.	Petroleum.		Resins and Bitumens, not specified.	Salt.		Silver.			
	Crude.	Refined.		Crude	Refined.	Ore.	Bullion, *Kg*.	Specie.	Jewelry, etc.
1895..............	2,312	29,239	76,087	2,136	117	19	45,299	$1,933,040	$ 51,452
1896..............	2	29,321	86,906	1,434	129	19	40,118	667,840	137,137
1897..............	1	18,088	92,591	493	231	423	57,933	20,851,320	189,959
1898..............	782	19,556	107,806	298	386	...	107,385	13,083,640	127,194
1899..............	2,146	25,970	112,392	506	885	...	54,358	13,483,160	214,175
1900..............	1,759	21,812	97,970	2,345	799	...	38,331	1,304,840	103,200
1901..............	*Nil.*	22,091	86,908	1,611	2,454	...	.....	....	85,000
1902..............	*Nil.*	23,344	101,950	1,378	1,077	...	.....	....	88,000
1903..............	*Nil.*	29,155	93,533	928	618	...	.....	....	94,400
1904..............	*Nil.*	30,209	97,462	1,899	1,955	...	.....	6,870,000	108,800

Year.	Sodium Salts.			Stone.		
	Carbonate.	Nitrate.	Sulphate and Sulphite.	Roofing Slate, *1000 Pieces*.	Building Stone, Including Marble and Alabaster.	Cut, Polished, etc.
1895..................	(*f*)	55,089	(*f*)	15,237	147,679	645,690
1896..................	(*f*)	42,857	(*f*)	15,435	161,298	922,147
1897..................	(*f*)	59,054	(*f*)	17,304	187,180	934,286
1898..................	(*f*)	106,252	(*f*)	16,948	178,249	861,015
1899..................	(*f*)	109,253	(*f*)	15,316	164,952	948,090
1900..................	25,569	39,346	10,577	12,836	171,126	872,400
1901..................	22,520	44,303	15,833	12,947	168,824	735,800
1902..................	12,831	33,319	14,316	13,715	163,210	848,800
1903..................	9,454	36,057	17,434	11,984	160,874	887,800
1904..................	15,049	36,970	18,715	11,362	160,421	822,800

Year.	Stone. (*Continued.*) Paving.	Stone. (*Continued.*) All Other Kinds.	Sulphur.	Tin. Block.	Tin. Manufactures.	Zinc. Spelter.	Zinc. Manufactures.
1895	134,838	780,863	4,576	1,051	$1,176	88,316	$ 54,210
1896	154,737	796,231	5,335	1,055	873	100,369	56,349
1897	153,504	773,531	6,041	347	839	100,228	90,749
1898	159,455	917,654	6,355	508	2,394	108,507	102,583
1899	150,993	834,528	6,769	659	1,511	101,244	109,762
1900	178,057	1,022,781	7,363	495	3,000	99,233	98,400
1901	148,176	729,196	6,722	299	1,200	106,656	98,600
1902	145,019	685,567	7,349	234	800	118,118	95,800
1903	128,715	696,338	10,324	838	1,000	119,988	156,200
1904	119,557	964,255	9,020	815	800	116,289	165,800

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays Etrangers.* (*b*) Pieces. (*c*) Guano only is reported. (*d*) Includes iron and steel filings. (*e*) Includes wrought tin plate. (*f*) Included under nitrate.

CANADA.

The mineral statistics of the Dominion of Canada as collected by the Geological Survey are summarized in the following tables:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (a)
(In metric tons or dollars.)

Year.	Arsenic.	Asbestos and Asbestic.	Barytes.	Cement—*Barrels.*		Chromite.	Clay Products.		
				Natural Rock.	Portland.		Fire Clay.	Pottery.	Terra-Cotta.
1896......	*Nil.*	11,113	132	70,705	78,385	2,124	764	$163,427	$83,855
1897......	*Nil.*	27,617	518	85,450	119,763	2,392	1,921	129,629	155,595
1898......	*Nil.*	21,577	971	87,125	163,084	1,833	608	135,000	167,902
1899......	52	22,938	653	131,387	255,366	1,796	543	185,000	220,258
1900......	275	27,797	1,213	125,428	292,124	2,335	1,129	200,000	259,450
1901......	630	36,477	592	133,328	317,066	1,274	3,609	200,000	278,671
1902......	726	36,657	994	127,931	594,594	900	2,486	200,000	276,241
1903......	725	37,902	1,055	92,252	627,741	3,509	2,394	200,000	405,796
1904(*g*)...	(*d*) 66	44,160	1,253	51,555	771,650	5,511	(*b*)	200,000	400,000
1905(*g*)...	(*b*)	61,946	3,049	14,184	1,346,547	7,781	(*b*)	(*b*)	64,892

Year.	Clay Products (*Continued*)	Coal.	Coke.	Copper. In Ore, etc.	Corundum.	Feldspar.	Gold. (*c*)	Graphite.	Grind-stones.
	Tiles and Sewer Pipe.								
1896...	$378,875	3,398,091	45,004	4,260	(*b*)	882	$2,754,774	126	3,368
1897...	389,250	3,434,756	55,042	6,032	(*b*)	1,270	6,027,016	395	4,147
1898...	406,717	3,784,532	79,453	8,048	(*b*)	2,268	13,775,420	(*b*)	4,476
1899...	386,546	4,467,021	91,444	6,838	(*b*)	2,721	21,261,584	1,188	4,091
1900...	456,525	5,087,060	142,521	8,588	3	288	27,908,153	1,743	5,024
1901...	498,115	5,648,208	331,537	17,155	403	4,852	24,128,503	2,004	4,155
1902...	551,965	6,524,180	455,353	17,598	697	6,871	21,336,667	993	5,835
1903...	592,970	6,933,107	509,115	19,357	880	12,633	18,843,590	660	5,023
1904...	653,894	6,814,755	493,246	19,497	934	10,057	16,400,000	410	4,582
1905...	(*f*)382,000	7,963,642	(*b*)	21,596	1,183	10,617	14,486,833	491	4,693

Year.	Gypsum.	Iron Ore.	Iron, Pig. All kinds.	Iron and Steel, Rolled.	Lead (In ore, etc.)	Lime.	Limestone for Flux.	Mangan-ese Ore.	Mica.
1896.......	187,778	83,359	61,012	76,244	10,975	$650,000	33,978	(*d*)112	$60,000
1897.......	217,340	45,989	52,612	78,253	17,695	650,000	28,365	(*d*) 14	76,000
1898.......	198,864	52,917	69,853	101,748	14,469	650,000	30,759	45	118,375
1899.......	221,821	67,678	93,367	112,412	9,914	800,000	47,006	1,434	163,000
1900.......	228,656	110,654	87,594	102,301	28,648	800,000	48,040	27	166,000
1901.......	266,476	284,477	248,859	113,799	23,537	830,000	153,645	(*d*) 399	160,000
1902.......	301,165	366,431	324,617	164,069	10,411	892,000	266,290	(*d*) 156	135,904
1903.......	285,242	239,715	270,182	131,588	8,226	860,000	251,649	83	177,857
1904.......	309,221	317,387	275,367	(*b*)	17,241	(*h*)	182,074	(*d*)112	152,170
1905.......	395,453	263,113	479,067	(*b*)	25,391	(*h*)	309,995	(*d*) 20	168,043

Year.	Mineral Paints. (Ochers)	Natural Gas.	Nickel. (In ore, etc.)	Petroleum, Crude. *Barrels.* (*e.*)	Phosphate (Apatite).	Platinum.	Pyrites.	Salt.	Sand and Gravel Exports.
1896......	2,142	$276,301	1,541	726,822	517	$750	30,580	39,872	203,865
1897......	3,542	325,873	1,813	709,857	824	1,600	35,291	46,574	138,737
1898......	2,019	322,123	2,502	758,391	665	1,500	29,223	51,828	150,520
1899......	3,555	387,271	2,605	808,570	2,721	825	25,112	53,820	219,902
1900......	1,783	417,094	3,211	710,498	1,283	*Nil.*	36,308	56,284	179,185
1901......	2,025	339,476	4,167	623,392	937	457	31,982	53,901	178,953
1902......	4,494	195,992	4,849	530,624	776	190	32,304	58,462	144,932
1903......	5,683	202,210	5,671	486,637	1,205	(*b*)	30,822	56,644	322,703
1904......	3,562	247,370	4,786	552,575	832	(*b*)	29,980	62,411	362,803
1905......	4,632	314,249	8,565	634,095	1,180	(*b*)	29,713	41,171	332,972

Year.	Sand (Molding).	Silver—*Kg.* (In ore, etc.)	Slate.	Soapstone and Talc.	Stone.			
					Building Bricks.	Building Stones.	Flagstones.	Granite.
1896.........	5,205	99,699	$53,370	372	$1,600,000	$1,000,000	$6,710	$106,709
1897.........	4,975	172,891	42,800	142	1,600,000	1,000,000	7,190	61,934
1898.........	9,589	138,486	40,791	367	1,900,000	1,300,000	4,250	81,073
1899.........	12,448	106,116	33,406	408	2,195,000	1,500,000	7,600	90,542
1900.........	5,606	138,980	12,100	1,288	2,275,000	1,520,000	5,250	80,000
1901.........	13,337	172,292	9,980	235	2,400,000	1,650,000	4,575	155,000
1902.........	12,110	133,478	19,200	625	2,593,000	1,900,000	7,760	210,000
1903.........	3,318	99,489	22,040	898	2,832,000	1,975,000	6,688	200,000
1904.........	(*b*)	115,666	23,247	762	(*h*)	5,667,000	6,720	100,000
1905.........	(*b*)	185,533	21,568	454	(*h*)	6,095,000	7,650	209,955

(*a*) From *Reports Compiled by the Geological Survey of Canada.* (*b*) Not reported. (*c*) Gold values are calculated at the rate of $20.67 per oz. (*d*) Export (*e*) One barrel contains 35 imp. gal. (*f*) Sewer pipe only reported. (*g*) From preliminary unrevised reports. (*h*) Included under building stone.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (*a.*)
(In metric tons or dollars.)

Year. (*b*)	Abrasives.				Aluminum.		Antimony. (*c*)	Arsenic.	Asbestos. (*d*)
	Buhrstones Number.	Emery, (Wheels and Bulk)	Grindstones.	Pumice stone.	Manufactures.	Ingots, Sheets, Etc.			
1896..........	1,572	$18,840	$26,561	$3,721	$7,537	....	74	301	$23,900
1897..........	1,499	11,231	25,547	2,903	5,717	....	61	68	19,032
1898..........	889	15,478	22,217	3,829	7,102	....	71	132	26,389
1889..........	1,116	22,343	27,476	5,973	9,275	....	131	264	32,607
1900..........	1,250	44,882	34,382	5,604	12,543	....	90	105	43,455
1901..........	3,641	39,116	39,068	5,516	16,202	....	159	72	50,829
1902..........	1,854	23,946	40,838	7,254	30,496	....	229	48	52,464
1903..........	35	40,235	53,388	6,152	14,201	$ 13,930	393	135	75,465
1904..........	2	50,899	46,039	6,537	16,065	101,427	190	188	83,827
1905..........	1,931	55,230	49,247	8,447	28,418	154,569	85	122	116,836

Year.	Asphalt.	Brass. (*e*)	Cement.	Chalk and Whiting.	Clay Products.		
					Bricks and Tiles.	Clays.	Earthenware and China.
1896..........	3,843	$ 477,279	$ 355,029	$33,789	$ 42,485	$ 62,984	$ 575,493
1897..........	342	457,342	260,842	29,973	44,622	59,386	595,822
1898..........	6,006	560,014	365,624	35,099	36,263	72,795	675,874
1899..........	8,196	747,557	477,617	44,771	55,204	88,517	916,727
1900..........	2,825	853,599	513,770	46,787	58,454	122,965	959,526
1901..........	2,849	985,776	666,350	72,507	76,760	141,251	1,114,677
1902..........	3,426	1,017,294	863,646	53,473	89,332	140,521	1,275,093
1903..........	3,037	1,197,012	890,745	56,364	157,783	176,416	1,406,610
1904..........	7,093	1,257,117	1,014,713	65,840	259,421	144,706	1,611,356
1905..........	5,096	1,375,907	1,263,828	77,387	369,561	197,609	1,636,214

Year.	Coal.		Coal Tar. *Barrels.*	Coke.	Copper.		Copper. Sulphate	Explosives.	Flint and Stones.
	Anthracite (f)	Bituminous. (f)			Ingots, Pig and Scrap.	Manufactures.			
1896	1,427,940	1,586,230	18,467	55,882	39	$ 285,220	710	$136,818	287
1897	1,321,767	1,604,517	23,661	75,580	22	264,587	516	131,562	475
1898	1,324,856	1,735,576	26,702	122 499	476	786,529	738	141,731	389
1899	1,583,132	2,220,250	39,296	128,145	751	551,586	726	212,968	243
1900	1,500,542	2,512,334	50,484	170,405	519	1,090,280	752	247,511	280
1901	1,753,488	2,658,257	54,928	280,069	432	951,045	673	306,067	222
1902	1,498,773	3,208,005	55,376	242,298	801	1,281,522	711	423,982	186
1903	1,320,239	3,684,502	29,325	232,848	924	1,291,635	1,010	347,020	403
1904	2,064,444	4,230,436	55,172	200,590	960	1,094,183	795	418,916	554
1905	2,361,952	4,377,667	77,856	337,035	882	1,666,955	934	369.311	844

Year	Fuller's Earth.	Glass.	Gold and Silver.		Gravel and Sand.	Graphite.	
			Coin and Bullion. (g)	Manufactures.		Crude	Manufactures. (h)
1896	$1,834	$1,104,481	$5,226,319	$340,241	17,193	$2,865	$37,931
1897	1,552	1,139,764	4,676,094	296,143	19,330	1,406	38,537
1898	3,330	1,024,706	4,390,844	297,242	29,164	1,862	52,291
1899	3,418	1,343,058	4,705,134	342,320	27,477	4,979	57,824
1900	2,661	1,658,694	8,297,438	339,145	32.399	4.437	60,518
1901	3,147	1,584,922	3,537,294	367,857	35,744	2,357	75,536
1902	3,909	1,932,539	6,311,405	352,224	42,995	3,649	64,123
1903	4,169	2,084,451	8,976,797	434,273	83,047	2,870	69,676
1904	5,554	1,983,781	7,874,313	441,154	100,394	1,802	67,563
1905	4,967	1,948,969	10,308,435	502,305	77,402	2,499	75,288

Year.	Gypsum.		Iron and Steel.				Kainite
	Crude and Ground.	Plaster of Paris.	Pig and Scrap.	Slabs, Blooms, Bars, Etc.	Alloys of Iron.	Manufactures.	
1896	977	135	46,051	5,832	592	10,263,985	180
1897	482	440	33,442	2,566	387	10,133,379	206
1898	1,057	150	81,577	7,391	1,287	15,458,355	49
1899	310	225	69,819	5,640	1,053	18,874,092	30
1900	72	385	94,489	11,576	1,043	27,259,134	143
1901	289	228	59,033	10,659	1,372	25,686,154	88
1902	516	215	71,882	18,208	5,910	31,257,412	85
1903	1,007	286	129,641	17,896	5,762	39,350,068	259
1904	626	291	86,087	9,088	2,700	31,014,946	339
1905	2,972	3,595	90,698	14,420	11,738	29,847,298	306

Year	Lead					Lime.		Mineral Paints. (ochers)
	Pig and Scrap.	Bars and Sheets.	Litharge.	Manufactures.	Pigments and Zinc White.	Burned. *Barrels.*	Chloride of.	
1896	3,286	389	486	$ 50,722	5,314	10,239	1,565	526
1897	2,962	477	546	60,735	4,678	16,108	1,361	682
1898	4,012	1,008	519	63,179	5,754	12,850	1,765	965
1899	5,202	2,032	432	91,497	6,583	15,720	1,857	1,110
1900	2,829	703	415	104,736	6,661	12,865	1,967	1,122
1901	(i)3,871	739	505	107,260	4,647	19,657	1,605	1,031
1902	(i)5,548	844	590	120,020	7,071	24,602	1,806	1,148
1903	(i)4,471	523	632	134,151	8,715	31,108	2,104	1,459
1904	4,292	......	......	133,639	7,679	54,359	2,080	1,256
1905	2,589	800	811	147,177	9,695	98,676	2,507	1,417

Year.	Mineral Waters.	Nickel.	Petroleum Products—*Gallons.*		Platinum.	Potassium Salts.	
			Illuminating oil, Etc. Crude or Refined.	Paraffine wax and Candles.		Except Saltpeter.	Saltpeter.
1896	$ 32,674	$ 4,787	8,005,891	80	$ 6,185	292	598
1897	22,142	4,737	8,415,302	74	9,031	265	456
1898	33,314	5,882	9,074,311	75	9,781	244	627
1899	38,046	9,446	10,394,208	70	9,671	472	930
1900	30,343	6,988	9,633,647	35	57,910	733	602
1901	40,802	12,029	11,082,822	74	20,263	476	581
1902	91,871	15,448	13,220,005	123	19,357	771	690
1903	108,130	26,177	18,799,312	307	21,251	1,060	916
1904	136,583	14,682	24,521,115	228	28,112	1,151	898
1905	161,790	19,076	13,229,855	98	61,719	945	1,048

Year.	Precious Stones and Jewelry.	Quick-Silver.	Sal-Ammoniac	Salt.	Silex.	Sodium Salts except Chloride.	Stone, Building.
1896	$ 675,420	35	52	96,493	149	12,326	$54,130
1897	506,728	35	69	103,337	116	13,938	38,714
1898	743,607	27	38	96,962	141	16,026	28,495
1899	923,837	47	53	88,397	179	20,742	48,040
1900	732,675	39	60	92,823	182	16,748	64,533
1901	1,279,617	64	76	103,402	162	18,631	46,078
1902	1,497,321	44	78	114,629	199	17,133	99,074
1903	1,977,359	75	114	112,188	159	18,887	87,866
1904	2,075,675	69	93	103,635	252	25,118	93,778
1905	2,315,889	47	143	97,723	405	26,219	102,817

Year.	Stone. (*Continued*).				Sulphur.	Tin and Tinware.	Zinc.	
	Lithographic Stones.	Marble.	Manufactures.	Slate.			Spelter.	Manufactures.
1896.......	$ 4,964	$90,065	$51,499	24,176	3,145	$1,237,684	918	$6,290
1897.......	6,360	77,150	34,026	21,615	3,932	1,274,108	542	5,145
1898.......	7,791	95,894	41,240	24,907	17,248	1,550,851	1,595	10,503
1899.......	6,223	101,879	60,148	33,100	11,121	1,372,813	852	14,661
1900.......	6,294	94,017	57,039	53,707	9,584	2,418,455	1,304	1,475
1901.......	9,584	96,159	66,639	72,187	10,827	2,339,109	931	6,882
1902.......	12,272	130,424	72,397	72,601	11,180	2,293,958	1,582	6,683
1903.......	8,461	153,481	78,629	84,437	11,077	2,712,186	1,209	9,754
1904.......	17,981	181,511	102,829	86,057	8,786	2,389,557	1,540	14,092
1905.......	13,683	145,466	150,160	93,228	10,633	2,791,757	1,721	11,912

EXPORTS OF DOMESTIC MINERAL PRODUCE FROM THE DOMINION OF CANADA. (*a*).

(In metric tons or dollars).

Year (*b*).	Antimony Ore.	Asbestos.	Clay Products.		Cement.	Chromite.	Coal.	Coke.
			Bricks. *Thousands*	Clay. Mfres. of				
1896...	(*i*)	8,700	1,216	$496	$1,035	(*k*)2,238	930,181	17
1897...	(*i*)	9,954	906	796	1,332	(*k*)1,911	1,000,061	1,692
1898...	1,118	16,718	276	343	609	(*k*)1,527	981,963	3,275
1899...	(*i*)	13,176	93	339	2,789	(*k*)1,369	1,035,245	4,024
1900...	6	16,483	342	215	2,274	(*k*) 334	1,489,139	12,558
1901...	219	24,242	728	761	3,554	(*k*)2,049	1,713,737	60,129
1902...	13	30,011	669	414	1,359	(*k*) 672	1,649,278	52,873
1903...	128	27,823	2,083	109	9,735	(*b*) 658	1,796,689	39,616
1904...	87	31,444	971	36	5,467	(*b*)2,103	1,494,106	61,750
1905...	340	37,320	670	2,755	5,430	(*b*)3,702	1,465,809	116,387

Year.	Coin and Bullion. (*m*)	Copper (*e*)	Explosives.	Fertilizer.	Glass and Glassware	Gold. Quartz, Dust, etc.	Graphite	Grindstones
1896...	207,532	1,622	92,763	$36,137	$6,881	$1,099,053	123	$18,853
1897...	327,298	4,596	76,578	36,584	7,208	2,804,101	78	15,760
1898...	1,045,923	6,319	74,305	46,864	7,494	3,387,953	348	18,785
1899..	1,101,245	3,843	115,065	51,224	11,788	3,272,702	662	18,619
1900...	1,670,068	6,274	155,764	51,410	11,016	14,148,543	1,742	22,196
1901...	1,978,489	11,954	240,535	37,706	13,574	24,445,156	1,246	38,304
1902...	1,669,420	13,789	248,434	61,831	11,587	19,668,015	783	21,878
1903...	619,963	13,445	254,605	116,474	14,065	16,437 528	530	14,169
1904...	2,465,557	20,279	212,124	177,193	21,452	18,715,539	269	12,676
1905...	1,844,811	17,431	184,531	229,212	16.163	15.208 380	201	27 985

Year.	Gypsum.		Iron and Steel.			Lead. (*p*)	Lime.	Manganese Ore.
	Crude.	Ground.	Iron Ore.	Pig and Scrap.	Manufactures.			
1896 ..	182,266	$23,332	(*n*) 2,675	....	....	9,211	$ 76.451	500
1897 ..	163,829	18,710	(*n*) 3,056	....	....	13,636	56,720	74
1898 ..	163,660	2,587	(*n*) 1,975	....	....	19,944	48,307	7
1899 ..	148,565	7,611	(*n*) 2,881	$ 90,505	$ 615,906	15,445	64,112	24
1900 ..	211,792	2,622	(*n*) 5,012	411,491	1,013,672	8,998	77,325	57
1901 ..	156,080	25,472	(*n*) 54,208	256,250	1,176,711	29,747	83,439	33
1902 ..	243,629	10,150	(*n*) 478,503	1,262,285	1,198,476	13,890	111,910	500
1903 ..	271,899	7,947	(*n*) 267,000	335,958	2,927,982	7,386	127,792	137
1904 ..	247,741	10,154	(*n*) 214,309	229,824	1,761,997	7,329	104,044	62
1905 ..	290,574	2,801	204,091	172,720	950,634	23.094	75,498	84

Year.	Mica.	Nickel in Ore, Matte, etc.	Petroleum, Crude and Refined.	Pyrites.	Salt. *Bushels.*	Silver—*Kg.* (In Ore, Matte, Etc.)	Stone. *All Kinds.*	Tin. Mfrs.
1896.......	253	3,174	18,241	6,706	3,007	78,016	51,941	$ 5,777
1897.......	217	3,415	1,831	14,219	4,702	127,440	23,700	2,764
1898.......	231	6,697	9,530	18,752	5,559	211,012	85,671	5,578
1899.......	538	6,546	4,268	11,707	5,209	137,400	110,290	3,159
1900.......	490	6,122	6,758	13,507	15,151	71,015	217,880	3,472
1901	444	4,327	19,942	22,146	56,461	125,110	182,342	14,481
1902	452	1,762	2,478	24,088	21,778	114,610	230,798	26,524
1903	632	4,098	413	16,762	7,959	100,861	212,097	90,953
1904	393	6,456	1,208	15,582	42,662	99,472	114,192	76,796
1905.......	461	5,431	6,441	20,473	5,663	112,076	78,791	37,535

(*a*) From Tables of the *Trade and Navigation of the Dominion of Canada.* (*b*) Fiscal year ending June 30. (*c*) Includes regulus and salts of antimony. (*d*) Asbestos in any form except crude, and all manufactures of. (*e*) Includes manufactures. (*f*) Includes coal dust. (*g*) Coin, gold and silver, except U. S. silver coin. (*h*) Includes black lead, and crucibles (clay or graphite). (*i*) Includes Canadian lead ore refined in the United States. (*k*) Calendar year. (*l*) Fine copper contained in ore, matte, regulus, etc. (*m*) Of foreign production. (*n*) Includes chromic iron ore. (*p*) Lead contained in ore, etc.

CHILE.

Until those of 1903, no statistics of Chile's mineral production were available. The following statistics are compiled from the first publication of this nature, prepared by the Sociedad Nacional de Mineria, Santiago, 1905:

MINERAL PRODUCTION OF CHILE. (*a*)
(In metric tons).

Year.	Borax.	Coal	Cobalt Ore.	Copper.	Gold Kg.	Guano.	Io-dine.	Mangan-ese Ore.	Salt.	Silver. *Kg.*	Sodium Nitrate.	Sul-phur.
1894....		(*c*)	(*d*)	23,274	1.659	(*f*)	(*e*)	47,994	......	144,750	1,103,033	832
1895....		(*c*)	(*d*)	22,387	1,789	(*f*)	(*e*)	24,075	1,010	136,877	1,260,446	931
1896....		(*c*)	(*d*)	23,649	1,634	(*f*)	(*e*)	26,151	2,434	150,480	1,158,088	940
1897....		(*c*)	(*d*)	21,128	1,538	(*f*)	(*e*)	23,529	5,867	140,732	1,148,696	664
1898....	(*b*)	(*c*)	(*d*)	26,331	2,037	(*f*)	(*e*)	20,851	6,684	131,995	1,283,563	1,256
1899....	86,892	(*c*)	(*d*)	25,719	2,060	(*f*)	274	40,931	9,937	129,503	1,389,823	989
1900....		(*c*)	(*d*)	27,715	1,975	(*f*)	302	25,715	9,879	73,071	1,460,100	2,472
1901....		(*c*)	(*d*)	30,155	1,100	(*f*)	269	18,480	10.099	70,237	1,273,800	2,516
1902....		(*c*)	(*d*)	27,066	1,286	(*f*)	242	12,990	9,532	57,418	1,400,408	2,636
1903....	16,879	827,112	285	29,923	1,425	11,134	157	17,110	16,264	39,012	1,444,920	3,441

(*a*) From *Estadistica Minera de Chile en* 1903. (*b*) The combined output of the years 1894 to 1902 inclusive. (*c*) The combined output of Chile up to the end of 1902 is estimated at 20,650,000 tons. (*d*) The combined output of Chile up to the end of 1902 is estimated at 5941 tons. (*e*) Not reported. (*f*) The combined output of Chile up to the end of 1902 is estimated at 296,545 tons, valued at 10,355,212 pesos ($3,779,652).

MINERAL EXPORTS OF CHILE. (*a*)
(In metric tons.)

Year.	Borate of Lime.	Coal.	Cobalt Ore.	Copper Matte.	Copper and Silver Matte.	Copper, Silver and Gold Matte.	Copper Ore.	Copper and Silver Ore–*Kg.*	Copper Silver and Gold Ore.	Copper Ingots.	Gold Bullion, *Kg.*
1894	6,700	205,201	4.6	342	1,508	2.5	11,106	90.3	460	19,640	1,475.4
1895	4,425	195,115	13.4	417	664	15.3	6,963	84.4	2,012	20,042	1,184.5
1896	7,486	204,858	(*b*)	2,528	1,059	7.6	6,159	62.3	29,542	20,592	1,061.3
1897	3,154	243,968	6.0	2,519	904	(*b*)	3,396	161.8	(*b*)	19,011	1,131.7
1898	7,028	282,663	18.2	3,079	419	17.8	20,301	87.0	5,733	20,600	1,630.5
1899	14,951	241,995	55.	1,710	1,094	93.0	35,854	184.0	12,000	17,311	1,625.0
1900	13,177	325,042	26.8	4,838	1,918	241.8	20,213	238.5	360	20,340	1,871.1
1901	11,457	(*d*)	76.0	2,905	1,779	208.0	15,921	119.0	60	24,480	637.0
1902	14,327	(*d*)	464.0	2,094	(*b*)	220.0	22,622	133.0	2,000	21,197	762.0
1903	16,879	200,000	284.5	2,689	864	863.6	17,961	89.8	440	22,196	207.5

Year.	Gold Ore.	Iodine.	Lead and Silver in Bars.	Manganese Ore.	Mineral Specimens.	Silver Ore.	Silver and Gold Ore.	Silver Ingots. *Kg.*	Silver Lead Ore.	Silver-Sulp'de Ore.	Sodium Nitrate.
1894	192	323	87	47,994	$ 1,150	370	56	153,723	15	127	1,081,337
1895	270	144	93	24,075	2,800	2,137	113	148,747	21	99	1,220,427
1896	367	206	594	26,151	700	2,750	666	151,226	*Nil.*	160	1,111,757
1897	64	243	369	23,529	20,300	984	260	143,541	6	183	1,057,640
1898	8	235	13	20,851	1,400	284	269	139,756	12	290	1,294,227
1899	12	304	171	40,931	64,521	302	370	75,899	32	339	1,380,718
1900	129	318	14	25,715	3,550	225	217	45,623	1	172	1,465,935
1901	66	385	455	18,480	(*b*)	6,166	196	46,164	(*b*)	264	1,291,958
1902	115	244	99	12,990	(*b*)	114	610	31,812	161	176	1,330,598
1903	57	387	*Nil.*	17,110	292	55	1,216	(*c*) 10,857	102	17	1,443,286

(*a*) From *Estadistica Minera de Chile en* 1903. (*b*) Not reported. (*c*) Contains 500 kg. of gold. (*d*) Not reported. There is no real exportation of Chilean coal for foreign consumption; that supplied to steam vessels is considered as exported.

FRANCE.

In the following tables are given the statistics of mineral and metal production in France and the French colonies, Algeria, New Caledonia and Tunis, together with the foreign commerce of France in mineral and metal products:

MINERAL AND METALLURGICAL PRODUCTION OF FRANCE. (a)

(In metric tons.)

Year.	Aluminum.	Antimony.		Arsenic Ore.	Asphaltum.	Barytes.	Bauxite.	Bitumen. (c)
		Ore.	Metal.					
1895	360	5,396	779	....	15,705	2,530	17,958	266,660
1896	370	5,675	969	....	17,717	2,791	33,820	225,784
1897	470	4,685	1,033	....	17,982	3,209	41,740	233,328
1898	565	4,433	1,226	....	18,832	2,763	36,723	229,108
1899	763	7,392	1,499	2,600	22,100	4,058	48,215	258,449
1900	1,026	7,843	1,573	4,705	25,228	3,635	58,530	266,474
1901	1,200	9,867	1,786	7,491	20,391	4,145	76,620	249,655
1902	1,355	9,715	1,725	5,372	....	4,323	96,900	258,295
1903	1,570	12,380	2,748	6,658	....	5,731	133,890	243,295
1904	1,650	9,065	2,116	3,117	....	6,944	75,640	227,177

Year.	Cement.	Clay Products.		Coal.			Copper.	
		Potter's Clay.	Fire Clay.	Bituminous.	Lignitic.	Peat.	Ore.	Metal.
1895	838,739	182,357	533,819	27,528,819	437,074	131,547	(b)	8,245
1896	934,624	246,677	291,690	28,750,452	439,448	130,207	106	6,544
1897	976,813	270,292	318,185	30,337,207	460,422	98,067	956	7,376
1898	1,072,025	260,362	295,913	31,826,127	529,977	104,265	382	7,834
1899	1,144,271	310,220	367,432	32,256,148	606,564	99,230	2,021	6,640
1900	1,147,670	331,396	329,561	32,721,562	682,736	95,630	3,031	6,446
1901	1,127,206	341,407	293,208	31,633,300	691,700	118,433	3,413	7,000
1902	962,930	4,541,359	295,341	29,365,047	632,423	109,941	828	6,300
1903	898,393	4,734,924	253,460	34,217,661	688,757	100,348	10,892	6,921
1904	903,632	4,968,936	220,409	33,502,394	665,572	95,716	2,756	6,900

Year.	Gold.	Gypsum.		Iron.				Lead.	
		Crude.	Calcined.	Ore.	Pig.	Wrought Iron.	Wrought Steel.	Ore. (d)	Pig. (e)
1895	$252,529	85,885	1,888,236	3,679,767	2,003,868	756,793	714,523	21,503	7,627
1896	217,308	264,187	1,429,550	4,069,390	2,339,537	828,758	916,817	19,042	8,232
1897	183,416	292,753	1,369,269	4,582,236	2,484,191	584,540	994,891	21,212	9,916
1898	177,435	303,531	1,449,384	4,731,394	2,525,100	766,000	1,174,000	23,342	10,920
1899	179,429	263,879	1,372,067	4,985,702	2,578,400	834,000	1,240,000	17,505	15,981
1900	134,904	192,916	1,405,845	4,676,740	2,714,298	672,172	1,226,537	24,276	15,210
1901	85,727	355,995	1,623,710	4,260,747	2,388,823	612,362	1,175,454	20,644	21,000
1902	(b)	219,487	1,572,687	5,003,782	2,405,000	639,600	1,245,800	22,634	19,000
1903	(b)	162,766	1,468,830	6,219,541	2,840,517	598,910	1,305,709	23,080	23,258
1904	(b)	106,173	1,481,303	7,022,841	2,999,787	554,632	1,482,708	14,173	18,800

Year.	Lime.	Manganese Ore.	Millstones.	Mineral Paints. (Ochers.)	Nickel.	Phosphate Rock.	Pyrites.	Salt.
1895	1,821,215	30,871	33,396	33,082	1,545	526,784	253,416	871,312
1896	2,224,847	31,318	28,237	27,499	1,545	582,667	282,064	1,042,614
1897	2,201,428	37,212	32,175	32,299	1,245	535,390	303,488	948,003
1898	2,339,850	31,935	(*f*)38,929	33,780	1,540	568,558	310,972	999,283
1899	2,343,377	39,897	41,535	32,750	1,740	645,868	318,832	1,193,532
1900	2,377,110	28,992	41,103	33,080	1,700	587,919	305,073	1,088,634
1901	2,443,062	22,304	33,286	35,704	1,800	535,676	307,447	910,000
1902	4,796,807	12,536	34,504	34,770	1,600	543,900	318,235	863,927
1903	4,727,543	11,583	35,031	34,042	1,500	475,783	322,118	967,531
1904	4,583,522	11,254	37,409	34,945	1,500	423,521	271,544	1,153,754

Year.	Slate.		Stone.				Sulphur Ore. (*g*)	Zinc.	
	Roofing.	Slabs.	Building.	Limestone. (Flux)	Marble.	Paving Blocks.		Ore.	Metal.
1895	291,444	2,103	10,142,933	900,422	39,599	812,342	4,213	72,989	24,230
1896	283,352	1,148	10,089,845	721,296	119,168	677,213	9,720	81,346	35,585
1897	310 820	1,143	10,105,438	709,562	118,675	568,677	10,723	83,044	38,067
1898	311,911	1,318	9,989,416	695,501	124,161	568,483	9,818	85,550	37,155
1899	299,307	1,162	10,587,789	924,945	191,030	621,799	11,744	84,813	39,274
1900	290,204	1,325	9,974,347	1,040,805	154,414	659,125	11,551	67,059	36,305
1901	288,508	1,304	10,277,098	1,083,372	123,506	604,464	7,000	61,539	37,600
1902	320,098	1,410	10,725,607	628,272	118,894	554,854	8,021	57,982	36,300
1903	382,461	1,404	10,713,356	704,736	136,615	577,554	7,375	66,922	37,416
1904	382,435	2,136	10,515,909	734,502	118,654	568,943	5,447	52,842	41,600

(*a*) From *Statistique de l'Industrie Minerale.* (*b*) Not reported. (*c*) Includes pure bitumen, bituminous schist and sand, and asphaltic limestone. (*d*) Argentiferous lead ore. (*e*) Lead produced from native ores only. (*f*) Finished product. (*g*) Sulphur and limestone impregnated with sulphur.

MINERAL PRODUCTION OF ALGERIA. (*a*)
(In metric tons.)

Year.	Antimony Ore.	Clays.	Copper Ore.	Gypsum.		Iron Ore.	Lead-silver Ore.
				Crude.	Plaster.		
1895	307	(*b*)	364	2,800	34,136	318,416	178
1896	658	48,297	427	300	29,870	374,476	117
1897	781	67,180	289	350	29,120	441,467	145
1898	138	78,690	488	150	29,750	473,569	120
1899	200	88,600	472	200	31,800	550,921	389
1900	93	94,000	....	500	37,100	174,000	222
1901	....	119,195	(*c*)7,267	600	34,740	161,303	1,614
1902	39	122,850	1,955	600	35,500	525,012	26
1903	490	125,800	100	300	33,000	588,893	499
1904	160	125,410	1,804	350	38,420	468,737	511

Year.	Lime.		Marble.	Onyx.	Phosphate Rock.	Salt.	Sand and Gravel.	Zinc Ore.
	Hydraulic.	White.						
1895	20,000	11,890	1,112	1,764	157,886	25,758	20,400	13,967
1896	20,000	9,450	900	900	165,738	19,658	41,400	17,587
1897	20,425	9,215	1,660	364	228,141	23,222	80,860	32,269
1898	13,000	12,975	985	219	269,500	21,300	72,185	29,800
1899	12,000	13,645	225	217	324,983	17,378	72,760	42,970
1900	12,000	13,700	....	228	319,422	18,325	71,860	30,281
1901	12,000	15,000	....	294	265,000	18,518	86,727	26,913
1902	....	....	375	150	305,174	27,263	72,180	33,139
1903	....	....	700	67	320,834	26,329	46,720	43,313
1904	....	....	530	121	343,317	18,563	51,020	47,192

(*a*) From *Statistique de l'Industrie Minerale.* (*b*) Not reported. (*c*) Copper ore.

MINERAL PRODUCTION OF NEW CALEDONIA. (*a*)
(In metric tons.)

Year.	Chrome Iron Ore.	Cobalt Ore.	Copper Ore.	Nickel Ore.
1895	8,079	4,277	(*b*)	29,623
1896	20,186	(*c*)4,823	(*b*)	6,417
1897	3,949	3,200	2,200	26,464
1898	7,712	2,373	*Nil.*	74,614
1899	12,634	3,294	6,349	103,908
1900	10,474	2,438	2	100,319
1901	17,451	3,123	6,349	132,814
1902	10,281	7,512	3.720	129,653
1903	21,437	8,292	10	77,360
1904	42.197	8,964	*Nil.*	98,655

(*a*) From *Statistique de l'Industrie Minerale.* (*b*) Not reported. (*c*) Exports.

MINERAL PRODUCTION OF TUNIS. (*a*)
(In metric tons.)

Year.	Salt.	Lead ore.	Phosphate of lime.	Zinc ore.
1895	8,000	(*b*)	(*b*)	14,800
1896	5,500	(*b*)	1,000	12,100
1897	8,100	2,123	(*b*)	11,830
1898	7,300	2,375	(*b*)	21,477
1899	8,850	2,263	70,000	20,079
1900	9,160	6,864	178,000	16,596
1901	16,900	8,158	172,000	17,879
1902	21,600	12,892	264,930	18,400
1903	18,846	12,752	352,088	21,262
1904	23,600	16,800	455,197	27,200

(*a*) From *Annual General Reports*, by C. Le Neve Foster, and *Statistique de l'Industrie Minerale.* (*b*) Not reported.

MINERAL IMPORTS OF FRANCE. (*a*)
(In metric tons or dollars. 5 f.=$1.)

Year.	Alum.	Bitumen. (*f*)	Borax.	Bromides.	Cement.	Copper.		
						Coal and Coke.	Ore.	Ingot and Mfres.
1895	109	43,975	442	12	13,441	10,261,069	10,450	38,196
1896	41	30,954	255	13	14,395	10,180,449	8,584	46,830
1897	54	29,931	264	18	15,141	10,457,255	11,960	54,460
1898	27	20,385	139	30	11,290	10,445,090	8,779	52,976
1899	34	30,770	123	46	13,640	11,896,030	8,517	58,419
1900	23	39,598	111	10	13,612	14,601,981	9,766	61,638
1901	39	28,888	128	3	16,232	13,925,623	13,383	47,035
1902	36	26,053	141	3	15,720	13,137,720	17,862	54,484
1903	138	27,573	312	9	21,152	14,029,687	9,796	59,126
1904	370	17,178	3,113	17	21,702	13,936,475	9,942	69,183

Year.	Copper.		Cobalt Oxide.	Gold. Bullion and Specie.	Iron.				
	Sulphate.	Oxide.			Ore.	Pig.	Iron and Steel, Mfres. of.	Sulphate.	Oxide.
1895	24,404	24	5	$50,775,039	1,651,369	36,247	66,240	3,882	855
1896	33,803	22	5	60,167,745	1,862,043	18,323	48,423	3,086	897
1897	30,132	29	9	58,143,077	2,137,860	35,633	60,804	1,353	1,125
1898	30,897	52	9	39,881,575	2,032,240	(*b*)	47,325	896	1,021
1899	21,733	36	9	63,697,020	1,950,665	(*b*)	64,178	1,698	1,037
1900	22,820	84	9	90,408,723	2,119,003	149,755	118,152	1,589	1,022
1901	15,313	162	8	85,485,000	1,662,875	61,085	77,742	45	1,001
1902	22,273	111	10	88,091,400	1,563,334	38,521	60,697	17	1,051
1903	25,428	129	11	62,154,022	1,832,820	121,726	119,799	36	1,207
1904	30,856	142	69	133,737,561	1,738,514	135,252	125,709	319	1,151

Year.	Kaolin.	Lead.			Lime.		Manganese Ore.
		Ore.	Carbonate.	Pig, scrap and Mfres.	Common and Hydraulic.	Chloride of.	
1895	359	5,032	1,077	66,241	246,677	1,047	41,400
1896	38,703	5,569	892	79,752	283,707	2,033	61,600
1897	42,384	13,981	1,327	86,589	321,047	1,713	85,500
1898	40,352	14,377	1,376	74,902	346,000	1,288	100,243
1899	36,904	12,637	2,029	67,149	321,610	1,887	106,630
1900	39,842	19,772	1,739	70,857	399,092	1,215	120.790
1901	41,972	15,430	1,789	59,051	374,281	1,400	94,365
1902	41,165	13,121	2,223	58,694	359,210	2,130	85,629
1903	47,534	20,172	2,040	75,416	386,612	919	109,930
1904	50,465	25,731	2,221	76,198	403,679	1,679	105,652

Year.	Nickel.		Petroleum.	Phosphate Rock.	Plaster.	Platinum. Kg.	Potassium.	
	Ore.	Metal.					Chloride.	Chromate (h)
1895	10,303	252	258,700	139,600	2,412	926	3,524	2,875
1896	15,756	425	272,693	256,888	1,774	2,117	11,499	2,838
1897	17,441	316	288,671	313,608	1,869	1,069	11,630	2,852
1898	24,935	330	291,961	336,842	2,040	505	10,929	2,890
1899	28,620	286	306,078	242,021	2,260	817	13,335	3,147
1900	17,687	299	302,482	283,921	3,648	2,398	13,524	3,293
1901	39,497	252	225,962	275,285	2,844	1,857	13,299	2,784
1902	58,374	301	148,170	302,898	2,440	2,940	10,802	2,861
1903	13,933	427	(g)476,230	343,012	2.664	3,764	12,275	2,760
1904	20,698	313	(g)435,730	419,720	2,674	5,650	14,734	2,618

Year.	Potassium. (Cont'd)		Pyrites.	Quicksilver.		Salammoniac.	Salt.	Sodium.	
	Nitrate.	Carbonate		Ore.	Metal.			Hydrate.	Nitrate.
1895	775	796	67,930	23	178	9,923	17,528	1,021	8,624,200
1896	2,614	1,526	45,788	25	234	15,256	17,191	1,109	9,025,400
1897	1,309	1,769	69,470	24	248	27,454	32,917	1,378	8,105,400
1898	1,008	2,418	71,569	19	221	20,426	35,863	1,772	8,026,400
1899	1,015	2,779	109,696	21	276	12,210	37,970	1,494	9,341,600
1900	1,928	2,768	156,825	22	161	15,205	32,045	1,062	11,995,820
1901	757	2,520	205,617	23	205	9,268	32,347	869	10,526,400
1902	1,547	1,539	170,783	24	224	15,446	32,505	643	9,372,600
1903	1,530	3,019	205,322	20	220	12,462	48,556	781	10,810,775
1904	2,117	3,781	230,097	22	208	13,744	46,232	1,068	9,074,859

Year.	Sulphur.	Sulphuric Acid.	Superphosphate of Lime.	Tin.		Zinc.	
				Ore.	Metal.	Ore.	Metal.
1895	110,989	3,461	150,758	104	7,691	41,622	25,652
1896	111,515	3,995	185,602	7	8,400	50,899	33,459
1897	136,118	3,147	195,853	149	7,642	58,074	31,211
1898	130,289	4,666	178,569	357	9,247	60,481	32,342
1899	120,062	4,583	171,631	486	6,907	78,192	25,516
1900	133,531	4,254	143,437	512	7,324	66,178	33,144
1901	101,301	5,386	165,361	365	7,314	74,553	29,812
1902	85,839	7,793	116,093	748	8,575	69,451	36,564
1903	109,594	13,241	89,229	1,808	9,873	67,258	39,305
1904	148,547	11,212	72,921	1,344	9,352	88,083	35,737

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (a)

(In metric tons.)

Year.	Aluminum.	Antimony.		Cement.	Coal.	Copper.		Gold. Kg. (d)
		Ore.	Metal.			Ore. (c)	Metal	
1895	110	832	68	(b)	(b)	1,772	8,829	1,353
1896	793	736	74	242,247	1,044,820	1,261	10,494	2,193
1897	224	623	61	244,504	1,142,195	2,000	12,667	3,335
1898	192	616	101	241,150	1,320,616	1,783	14,350	1,812
1899	256	304	255	244,480	1,229,090	2,078	17,949	2,622
1900	324	154	336	232,577	1,201,210	9,197	16,791	883
1901	307	645	741	242,010	908,583	16,066	14,776	1,869
1902	748	595	666	210,590	910,760	20,489	14,423	1,517
1903	666	904	1,358	233,835	2,238,735	12,487	11,403	3.139
1904	664	1,191	720	260,686	2,384,928	14,258	12,663	1.537

Year.	Iron.				Pyrites.	Lead.		Manganese Ore.
	Ore.	Pig.	Bars.	Steel.		Ore.	Metal.	
1895	236,923	150,540	....	29,074	37,968	8,670	8,037	16,193
1896	238,430	195,212	24,721	44,795	44,232	8,597	10,856	10,913
1897	299,589	108,645	39,894	45,809	54,367	12,007	10,364	19,464
1898	236,169	162,991	27,424	47,278	60,406	10,216	3,663	12,229
1899	291,346	153,792	29,112	33,584	53,395	3,909	1,163	12,289
1900	371,799	114,361	18,763	19,535	64,530	2,345	958	8,392
1901	258,925	96,463	25,220	56,347	52,952	3,490	718	5,289
1902	422,677	213,081	23,828	121,932	63,920	2,414	648	1,948
1903	714,173	196,444	40,533	215,737	119,173	2,313	13,048	717
1904	1,219,149	191,819	40,374	246,738	40,833	1,860	13,467	1,392

Year.	Mill-stones. *Number.*	Nickel Refined.	Phosphate. Rock.	Plaster.	Silver. *Kg.* (*e*)	Tin. (metal)	Zinc.	
							Ore.	Spelter, Sheets and Scrap.
1895	....	408	....	....	13,567	650	61,291	5,849
1896	196,685	490	48,719	89,952	9,849	744	62,415	10,485
1897	158,979	498	69,188	107,823	5,374	651	79,909	10,977
1898	203,584	526	93,742	106,790	1,886	587	60,664	16,995
1899	112,620	280	70,517	112,520	....	666	76,104	14,958
1900	65,436	599	89,135	108,387	15,470	716	54,663	12,712
1901	52,383	1,031	81,405	101,063	16,745	438	42,995	15,022
1902	45,647	397	62,375	110,270	17,184	654	47,724	16,158
1903	11,557	720	72,252	131,245	43,690	1,994	62,731	12,657
1904	14,479	906	78,612	139,551	23,105	2,300	57,780	19,063

(*a*) From *L'Economiste Francais* (representing the *Commerce Special*) except for last two years, which were from *Tableau Général du Commerce et de la Navigation*. (*b*) Not reported. (*c*) Includes matte. (*d*) Gold and platinum in ore sheets, leaves or threads. (*e*) Silver in ore, sheets, leaves, wire, etc. (*f*) Includes bitumen, bituminous schist and sands and asphaltic limestone. (*g*) Crude and refined. Transposition from hectoliters to tons was performed by assuming specific gravity of petroleum to be .9. (*h*) Includes chromate of soda

GERMANY.

The mineral production, imports and exports of the German Empire are given in the following tables in metric tons unless otherwise specified, or in dollars, on the basis of four marks to the dollar.

PRODUCTION. (a)

Year.	Alum.	Aluminum Sulphate.	Arsenic.		Asphaltum.	Bora-cite.	Cadmium. Kg.	Coal.	
			Ore.	Salts.				Bituminous.	Lignitic.
1896.....	3,430	34,370	3,691	2,637	61,552	184	10,000	85,690,233	26,780,873
1897.....	2,995	37,053	3,777	2,989	61,645	198	15,531	91,054,982	29,419,503
1898.....	4,069	35,366	3,527	2,679	67,649	230	14,943	96,309,652	31,648,898
1899.....	3,358	37,693	3,834	2,423	74,770	183	13,608	101,639,753	34,204,666
1900.....	4,355	44,372	4,379	2,415	89,685	232	13,553	109,290,237	40,498,019
1901.....	4,145	46,807	4,035	2,549	90,193	184	13,144	108,539,444	44,479,970
1902.....	4,108	47,905	3,959	2,828	88,374	196	......	107,473,933	43,126,281
1903.....	3,934	49,727	4,369	2,768	87,454	159	16,565	116,637,765	45,819,488
1904.....	3,850	55,881	4,390	2,829	91,736	135	25,245	120,815,503	48,635,080
1905.....	4,127	52,892	4,887	2,535	103,006	183	24,568	121,298,167	52,498,507

Year.	Cobalt, Nickel and Bismuth Ores.	Copper.				Gold.	Graphite.
		Ore.	Matte. (b)	Ingots.	Sulphate.		
1896................	4,087	717,347	598	29,319	6,046	$1,652,736	5,248
1897................	3,355	700,619	315	29,408	5,549	1,848,114	3,861
1898................	3,157	702,781	62	30,695	4,352	1,891,974	4,593
1899................	1,270	733,619	95	34,634	5,142	1,731,153	5,196
1900................	4,495	747,749	4,207	30,929	5,076	2,030,200	9,248
1901................	10,479	777,339	365	31,317	5,192	1,830,835	4,435
1902................	12,433	761,921	447	30,578	4,997	1,770,361	5,023
1903................	14,607	772,695	583	31,214	5,200	1,709,223	3,720
1904................	14,016	798,214	641	30,264	6,584	1,819,538	3,784
1905................	10,848	793,498	1,635	31,717	6,988	2,613,675	4,921

Year.	Iron and Steel.					Lead.		
	Iron Ore.	Pig Iron. (c)	Castings.	Steel.	Sulphate. (d)	Ore.	Pig.	Litharge.
1896................	14,162,335	6,372,575	1,384,008	6,021,240	9,788	157,504	113,793	3,930
1897................	15,465,980	6,881,466	1,473,211	6,248,141	10,351	150,178	118,881	3,341
1898................	15,901,263	7,312,766	1,597,434	6,941.278	10,422	149,311	132,742	3,857
1899................	17,989,635	8,153,133	1,776,878	7,532,524	10,931	144,370	129,225	3,562
1900................	18,964,294	8,520,540	1,812,603	7,377,275	10,913	148,257	121,513	3,088
1901................	16,570,182	7,880,087	1,520,617	7,033,438	11,148	153,341	123,098	4,101
1902................	17,963,591	8,529,900	1,575,525	8,317,231	......	167,855	140,331	4,197
1903................	21,230,650	10,017,901	1,721,781	9,226,898	12,243	165,991	145,319	4,428
1904................	22,047,393	10,058,273	1,879,879	9,239,302	13,585	164,440	137,580	4,332
1905................	23,444,073	10,875,061	2,045,477	10,309,690	12,949	152,725	152,590	3,786

Year.	Magnesium Salts.		Mangan-ese Ore.	Nickel. (e).	Petro-leum.	Potassium Salts.				
	Chloride.	Sul-phate.				Chloride.	Kainite. (f)	Sul-phate.	Potassium and Magnesium Sulphate.	Unspeci-fied.
1896........	17,525	27,161	45,062	1,391	20,395	174,515	877,885	19,682	4,623	902,707
1897........	18,014	35,072	46,427	1,464	23,303	168,001	992,389	13,774	7,812	953,798
1898........	19,819	30,295	43,354	1,691	25,989	191,347	1,103 643	18,853	13,982	1,105,212
1899........	21,370	39,540	61,329	1,747	27,027	207,506	1,108,159	26,103	9,765	1,384.972
1900........	19,397	48,591	59,204	1,989	50,375	271,512	1,227,873	30,853	15,368	1,822,758
1901........	21,018	46,714	56,691	2,207	44,095	294,666	1,498,569	37,394	15,612	2,036,325
1902........	19,658	39,262	49,812	2,196	49,725	267,512	1,322,623	28,278	18,147	1,962,384
1903........	22,990	37,844	47,994	2,637	62,680	280,248	1,557,243	36,674	23,631	2,073,720
1904........	25,730	39,412	52,886	3,063	89,620	297,238	1,905,893	43,959	29,285	2,179,471
1905........	......	58,758	51,463	3,317	78,869	3 0,914	2,317,829	47,994	34,032	2,725,654

Year.	Pyrites.	Salt. Rock.	Salt. Evaporated.	Silver and Gold Ore.	Silver. *Kg.*	Sodium Sulphate.	Sulphur.	Sulphuric Acid.
1896	129,168	758,866	547,486	11,320	428,429	71,958	2,263	664,741
1897	133,302	763,412	543,272	9,708	448,068	68,822	2,317	702,445
1898	136,849	807,792	565,683	14,702	480,578	69,111	1,954	754,151
1899	144,623	861,123	571,058	13,506	467,590	79,062	1,663	813,141
1900	169,447	926,563	587,464	12,593	415,735	90,468	1,445	829,376
1901	157,433	985,050	578,751	11,577	403,796	76,066	963	835,000
1902	165,225	1,010,412	572,846	11,724	430,610	90,742	.....	894,409
1903	170,867	1,095,541	598,394	11,467	396,253	83,087	219	928,190
1904	174,782	1,079,868	621,064	10,405	389,827	75,171	209	963,384
1905	185,384	1,165,473	612,062	9,628	399,775	68,454	205	1,173,680

Year.	Tin. Ore.	Tin. Block.	Tin. Chloride.	Uranium and Tungsten Ores.	Zinc. Ore.	Zinc. Spelter.	Zinc. Sulphate.
1896	88	826	.....	41	729,942	153,100	4,811
1897	55	929	.....	38	663,850	150,739	5,488
1898	51	993	......	50	641,706	154,867	6,104
1899	72	1,481	. ...	50	664,536	153,155	7,117
1900	80	2,031	(*g*) 143	43	639,215	155,790	6,027
1901	82	1,464	(*g*) 135	43	647,496	166,283	5,552
1902	104	2,779	.....	31	702,504	174,927	.....
1903	110	3,065	1,064	35	682,853	182,548	5,994
1904	99	4,216	816	23	715,732	193,058	6,185
1905	123	5,233	811	26	731,281	198,208	5,896

(*a*) From the *Vierteljahrshefte zur Statistik des Deutschen Reichs.* (*b*) Includes black copper. (*c*) Includes blast-furnace, bessemer and basic pig, castings from the blast furnace, ferro manganese and spiegeleisen, forge pig and scrap. (*d*) Contains a small quantity of copper and iron sulphate mixed. (*e*) Includes nickeliferous by-products, metallic bismuth, and uranium compounds. (*f*) Compound of potassium chloride and magnesium sulphate. (*g*) Includes nickel sulphate.

IMPORTS. (*a*)

Year.	Aluminum, Refined and Crude.	Aluminum, Nickel, Wares, Etc.	Ammonium Sulphate.	Antimony.	Antimony and Arsenic Ores.	Asbestos. Crude.	Asbestos. Mfr.
1896	....	335	32,061	....	....	....	....
1897	....	417	33,113	....	....	....	....
1898	....	459	30,254	....	....	....	....
1899	....	495	28,868	....	....	....	....
1900	943	483	23,105	1,461	1,291	6,850	152
1901	1,090	438	44,408	1,494	1,098	5,500	161
1902	1,100	418	42,252	1,495	1,231	3,415	113
1903	1,155	455	35,168	2,281	1,741	5,727	147
1904	2,422	480	35,166	2,003	1,687	5,251	208
1905	3,252	488	48,005	1,680	567	7,830	206

Year.	Asphalt.	Bituminous Rock.	Barium Chloride.	Barytes. (*b*)	Brass, Tombac and Scrap.	Borax.	Bauxite.	Calcium Carbide.	Cement.
1896	....	....	....	....	....	....	....	....	32,394
1897	....	....	....	....	....	....	....	....	42,364
1898	....	....	....	....	....	....	....	....	53,519
1899	61,534	....	....	....	....	....	....	....	63,388
1900	80,765	48,986	3,062	7,282	2,214	2,403	29,383	7,703	79,303
1901	62,299	41,733	1,768	5,764	1,709	2,537	24,113	9,526	87,262
1902	88,536	36,791	2,135	5,040	1,192	2,057	26,698	11,287	52,018
1903	94,377	40,873	2,374	5,534	1,970	2,567	22,316	14,081	49,870
1904	85,049	38,812	2,428	6,742	3,174	2,603	27,849	14,840	60,188
1905	82,461	64,196	2,114	7,981	3,094	2,802	39,137	17,256	148,118

Year.	Chalk (*d*), Crude White.	Chrome Ore.	Clay Products. Tile, Brick and Building Stone.	Clay Products. Kaolin, Feldspar and Fire Clay.	Clay Products. Porcelain.	Coal. Bituminous.	Coal. Lignitic.	Coke.
1896	....	....	140,133	192,807	806	5,476,753	7,637,503	393,881
1897	....	....	151,455	207,155	812	6,072,029	8,111,076	435,161
1898	....	....	(*c*)186,183	208,186	788	5,820,332	8,450,149	332,579
1899	....	....	(*c*)176,704	235,233	704	6,220,489	8,616,751	462,577
1900	63,929	18,728	(*c*)180,094	265,289	765	7,384,049	7,960,313	512,690
1901	29,611	18,222	141,335	249,180	737	6,297,389	8,108,943	400,197
1902	26,408	10,152	130,470	229,556	698	6,425,658	7,882,010	362,488
1903	33,362	13,919	143,536	255,083	746	6,766,513	7,962,123	432,819
1904	32,581	18,132	149,073	296,073	825	7,299,042	7,669,099	550,302
1905	(*f*)35,529	11,998	175,235	321,859	750	9,399,693	7,945,261	713,619

Year.	Peat.	Briquettes and Peat Coke.	Cobalt and Nickel Ore.	Copper.					Copper and Brass Manufactures.		
				Ore.	Crude.	Bars and Sheets.	Coin and Scrap.	Sulphate.	Fine.	Wire.	Coarse Wares, Cartridges, Etc.
1896...	......	....	......	....	56,115	401	3,170	.....	797	127	582
1897....	......	....	......	....	67,573	400	4,199	......	846	80	612
1898....	......	....	......	....	73,291	450	4,720	......	881	55	546
1899....	......	....	......	....	70,091	610	4,992	......	962	79	565
1900....	19,807	137,153	13,032	10,930	83,503	906	4,603	2,369	1,007	76	504
1901....	15,102	92,037	12,186	4,614	58,620	786	4,536	1,211	834	91	439
1902....	16,696	81,854	15,551	14,630	76,050	540	4,369	2,499	866	124	404
1903....	14,640	84,635	36,927	13,714	83,261	568	5,533	1,691	977	185	615
1904....	9,071	125,477	14,555	7,949	110,231	719	6,440	1,735	966	108	972
1905....	11,439	191,753	39,590	10,137	102,218	927	6,942	2,180	1,085	192	723

Year.	Cryolite.	Glass. (All Kinds).	Gold.		Gold and Silver.	
			Bullion.	Specie.	Scrap, Pagament. Kg.	Manufactures.
1896.........	....	2,823	$30,181,867	$23,488,931	39,395	$731,832
1897.........	....	2,057	23,253,269	14,350,000	36,664	750,000
1898.........	....	2,408	41,824,783	37,779,250	54,793	687,500
1899.........	....	2,337	34,250,242	31,870,250	44,463	865,000
1900.........	1,460	2,746	24,650,818	32,784,738	45,774	1,603,250
1901.........	1,249	2,561	28,631,472	32 517,702	32,158	1,436,500
1902.........	1,332	2,452	17,300,895	15,790,824	30,552	1,831,750
1903.........	1,082	10,946	39,141,995	27,711,530	29,609	2,003,250
1904.........	1,139	11,234	52,068,157	56,034,482	36,769	1,791,500
1905.........	1,143	12,515	31,743,082	30,588,776	40,493	1,853,250

Year.	Gold, Silver and Platinum Ores.	Graphite.	Gypsum.	Infusorial Earth.	Iodine.	Iron and Steel.		
						Ore.	Pig.	Scrap.
1896.........	6,024	13,718	....	....	94	2,586,706	322,502	14,679
1897........	8,927	17,366	....	....	164	3,185,644	423,127	37,957
1898.........	7,841	20,269	....	....	216	3,516,577	384,561	23,328
1899........	7,597	23,400	....	....	191	4,165,372	612,652	63,141
1900........	9,153	22,495	7,571	240	236	4,107,840	726,712	100,383
1901........	8,764	17,374	7,622	208	266	4,370,022	267,503	26,363
1902........	6,585	19,392	8,177	235	220	3,957,403	143,040	31,950
1903........	4,386	20,953	8,328	330	320	5,225,336	158,347	59,980
1904........	5,960	23,533	9,550	215	272	6,061,127	178,256	52,421
1905........	6,225	26,143	11,247	249	377	6,085,196	158,700	40,254

Year.	Iron and Steel. (*Continued.*)								
	Wrought in Bars.	Wrought in Sheets and Plates.	Cast Iron.	Angles.	Blooms, Bars and Ingots.	Rails.	Wire.	All Other Mfrs.	Ferrous Sulphate.
1896.....	23,770	(*g*)	19,600	176	1,054	140	6,398	12,933	....
1897.....	29,467	(*g*)	24,627	1,081	1,038	774	5,609	18,135	....
1898.....	26,014	(*g*)	39,371	207	1,553	267	7,166	20,158	....
1899.	37,179	3,187	45,853	898	1,341	1,319	8,582	31,222	....
1900.....	37,809	3,629	45,348	827	2,778	343	8,711	28,913	752
1901.....	22,518	2,097	37,599	671	1,666	545	7,922	17,964	501
1902.....	24,579	1,600	22,350	184	1,549	136	7,245	30,054	807
1903.....	26,129	1,238	24,388	396	2,149	142	7,195	29,497	778
1904.....	26,066	1,165	23,516	683	9,556	310	7,868	36,685	765
1905.....	29,934	1,407	(*e*) 24,367	293	6,188	487	8,252	779,563	666

Year.	Lead.			Magnesite.	Manganese Ore.	Mineral Pigments.	Nickel.
	Ore.	Pig and Scrap.	Lead White.				
1896........................	....	33,016	579	....	63,870	....	951
1897........................	....	35,092	696	....	86,911	....	1,390
1898........................	....	47,497	822	....	130,711	....	1,467
1899........................	....	55,635	703	....	196,825	....	1,391
1900........................	51.338	70,252	698	13,920	204,420	12,107	1,712
1901........................	100,196	52,886	423	8,897	222,010	9,403	1,947
1902........................	71,078	39,006	357	12,237	204,647	7,719	1,458
1903........................	67,573	52,440	442	14,958	223,709	9,888	1,507
1904........................	83,807	61,388	622	15,877	255,760	10,494	1,712
1905........................	92,667	78,528	2,488	19,459	262,311	11,473	1,995

Year.	Ozokerite.	Petroleum Products. Illuminating Oil.	Petroleum Products. Lubricating Oil.	Phosphorus.	Phosphate Rock.	Potassium Salts. Chloride.	Potassium Salts. Cyanide.	Potassium Salts. Iodide.	Potassium Salts. Nitrate.	Potassium Salts. Carbonate.	Potassium Salts. Hydroxide.	Potassium Salts. Sulphate.
1896.		853,642	81,256	....	216,950	1,058	3	29	1,380	1,430	......	703
1897.	....	946,344	83,957	....	289,234	715	7	18	2,889	1,734	......	912
1898.	....	954,646	97,028	....	270,988	422	2	16	1,895	1,486	......	999
1899.	....	963,943	106,624	....	407,457	443	3	9	1,785	1,737	......	533
1900.	3,457	989,361	124,505	381	320,138	484	2	10	2,047	1,522	283	856
1901.	1,981	985,904	118,999	313	351,155	462	2	5	1,529	1,758	165	680
1902.	1,585	1,006,829	125,667	350	430,043	261	3	10	1,889	2,112	42	266
1903.	1,663	1,067,697	147,837	222	461,092	40	3	8	2,163	1,850	52	81
1904.	1,300	1,076,324	142,929	220	508,634	47	2	10	2,349	1,955	61	121
1905.	1,114	1,070,252	143,926	198	501,048	223	3	30	2,156	1,693	24	131

Year.	Pumice-stone.	Pyrites.	Quick-silver.	Salt.	Silica, Sand, Marl, Etc.	Silver. Bullion. *Kg.*	Silver. Specie. *Kg.*	Slag and Slag Wool.
1896........	....	343,852	(*e*)	22,908	253,905	127,753	....	680,251
1897........	....	356,869	(*e*)	(*e*)	228,241	147,034	....	670,224
1898........	....	376,817	560	21,957	239,708	104,770	....	685,118
1899........	....	437,732	572	22,040	279,089	89,930	....	892,764
1900........	2,154	457,679	555	21,738	386,028	167,432	36,857	974,947
1901........	2,336	488,633	651	23,901	264,686	197,855	40,885	733,931
1902........	2,070	482,095	648	26,404	305,235	282,774	39,936	831,282
1903........	2,697	519,317	674	20,118	249,475	293,117	38,675	877,394
1904........	3,000	503,503	691	18,743	303,419	338,875	35,189	846,738
1905........	3,240	552,184	729	20,726	320,839	428,485	34,721	888,665

Year.	Slag, Basic Slag, Ground.	Slate.	Sodium Salts. Soda, Calcined.	Sodium Salts. Nitrate (Chile Saltpeter.)	Sodium Salts. Sulphate.	Stassfurt Salts.
1896...........................	83,765	53,583	1,295	449,028	....	143
1897...........................	110,216	48,380	916	465,493	....	6
1898...........................	88,374	57,571	524	425,054	....	17
1899...........................	68,305	63,309	515	526,944	....	182
1900...........................	103,481	54,646	373	484,544	9,450	130
1901...........................	87,152	44,998	178	529,568	7,921	155
1902...........................	103,107	49,646	121	467,024	7,308	307
1903...........................	132,337	45,866	114	467,130	6,058	388
1904...........................	150,836	45,375	179	506,172	9,598	57
1905...........................	198,763	44,216	143	540,916	4,752	46

Year.	Stone. Rough or Simply Hewn.	Stone. Limestone.	Stone. Marble.	Stone. Grindstone Polishing & Whetstones.	Strontianite.	Sulphur.	Sulphuric Acid.	Superphosphate.
1896........	767,908	....	....	....	....	21,864	....	....
1897........	545,723	....	....	....	....	25,305	....	....
1898........	556,561	....	....	....	....	30,269	....	....
1899........	1,021,755	....	....	....	....	31,196	....	....
1900........	1,072,433	272,324	37,524	5,123	8,701	40,689	20,634	72,062
1901........	926.924	261,659	38,958	4,318	19,739	32,750	18,502	107,365
1902........	870,686	293,151	39,353	4,439	34,035	32,798	22,205	109,374
1903........	902,546	330,244	42,503	4,812	24,183	41,545	13,418	82,740
1904........	901,662	366,899	47,250	5,120	18,055	41,030	16,087	91,288
1905........	1,221,834	396,923	42,281	5,283	13,720	39,989	33,837	109,666

Year.	Tin. Crude.	Tin. Manufactures.	Zinc. Ore.	Zinc. Spelter.	Zinc. Drawn or Rolled.	Zinc. Manufactures.	Zinc. Zinc-white Zinc-gray Lithophon.
1896.......................	13,798	....	21,493	16,343	180	....	2,899
1897.......................	12,395	....	24,735	19,734	130	....	3,532
1898.......................	14,623	....	48,050	24,116	53	....	3,653
1899.......................	12,253	....	57,880	23,691	95	....	4,226
1900.......................	12,454	$170,750	68,982	24,263	145	$94,000	4,884
1901.......................	12,910	130,000	75,533	21,250	306	92,250	3,673
1902.......................	13,760	153,000	61,407	25,946	134	80,250	3,986
1903.......................	13,925	156,750	67,156	25,749	237	90,750	4,667
1904.......................	14,352	167,000	93,515	26,389	151	91,750	6,461
1905.......................	13,501	135,250	126,577	29,583	54	102,000	7,802

EXPORTS. (a)

Year.	Aluminum, Refined and Crude.	Aluminum, Nickel Wares, etc.	Aluminum Sulphate.	Ammonium. Carbonate and Chloride.	Ammonium. Sulphate.	Antimony and Arsenic Ores.	Antimony. Metallic.	Antimony. Salts.
1896	...	1,977	......	....	2,201	...	...	...
1897	...	1,899	......	....	2,623	...	...	...
1898	...	2,045	......	....	4,083	...	...	...
1899	...	2,312	......	....	1,553	...	...	...
1900	269	2,398	29,372	3,196	2,431	284	131	786
1901	282	2,270	31,171	3,196	9,842	283	76	826
1902	410	2,608	34,005	3,351	5,744	410	105	954
1903	353	2,865	28,513	2,778	5,592	427	83	872
1904	407	3,077	29,311	3,106	10,696	486	250	964
1905	1,192	3,476	34,776	3,579	27,589	287	218	1,097

Year.	Arsenic. Metallic.	Arsenic. White, etc.	Asbestos. Crude.	Asbestos. Manufactures.	Barytes. (b)	Barium. Chloride and and Salts of.	Barium. White.	Bauxite.	Borax.
1896	....	....	....	....	....	....	....	....	....
1897	....	....	....	....	....	....	....	....	....
1898	....	....	....	....	....	....	....	....	....
1899	....	....	....	....	....	....	....	....	....
1900	14	1,573	496	1,664	59,012	5,927	2,717	44	2,894
1901	28	1,534	638	1,680	67,526	6,803	2,765	137	2,563
1902	46	2,036	709	1,592	56,026	7,358	2,922	32	2,836
1903	32	1,903	513	1,696	72,455	8,417	3,187	19	2,779
1904	50	1,956	738	1,865	69,564	8,596	3,777	21	2,741
1905	40	1,753	1,173	2,566	81,134	9,550	4,382	6	2,720

Year.	Bromine.	Bromine Salts.	Calcium. Carbide.	Calcium. Chloride.	Cement.	Chalk. (d). Crude White.	Chromium. Ore.	Chromium. Alum.
1896	....	....	....	....	478,340	....	....	....
1897	....	....	....	....	524,557	....	....	....
1898	....	....	....	....	551,744	....	....	....
1899	....	....	....	....	580,255	....	....	....
1900	191	255	224	1,315	600,386	11,860	427	1,192
1901	228	249	275	888	560,612	14,134	581	1,299
1902	153	357	126	1,346	699,378	8,475	846	1,758
1903	155	435	335	1,831	742,381	12,211	37	1.921
1904	208	411	608	2,381	635,248	11,359	47	2,432
1905	156	634	709	2,831	675,664	13,081	43	2,507

Year.	Clay Products. Tile, Brick and Building Stone.	Clay Products. Kaolin, Feldspar and Fire Clay.	Clay Products. Porcelain.	Coal. Bituminous.	Coal. Lignitic.	Coke.	Peat.	Briquettes and Peat Coke.
1896	287,811	125,674	21,624	11,598,757	15,703	2,216,395	....	....
1897	233,229	121,535	21,687	12,389,907	19,112	2,161,886	....	....
1898	265,110	129,082	21,644	13,989,223	22,155	2,133,179	....	....
1899	224,819	143,406	23,110	13,943,174	20,925	2,137,985	....	....
1900	182,340	159,855	26,642	15,275,805	52,795	2,229,188	8,849	550,222
1901	166,834	128,174	27,649	15,266,267	21,718	2,096,931	11,588	529,765
1902	210,337	136,928	29,142	16,101,141	21,766	2,182,383	13,410	697,799
1903	225,878	145,829	32,337	17,389,934	22,499	2,523,351	16,986	895,145
1904	190,378	161,673	38,981	17,996,726	22,135	2,716,855	14,830	917,526
1905	164,900	174,130	40,219	18,156,998	20,118	2,761,080	16,009	936,694

Year.	Cobalt and Nickel Ores	Copper. Ore.	Copper. Bars, Sheets and Wire.	Copper. Crude.	Copper. Scrap and Coin.	Copper. Sulphate.	Copper. Manufactures.
1896	....	....	....	5,996	2,968	....	....
1897	....	....	....	7,183	2,164	....	....
1898	....	....	....	6,972	3,636	....	....
1899	....	....	....	7,061	5,217	....	....
1900	186	25,686	9,787	5,505	5,455	1,881	1,271
1901	96	26,678	7,700	5,097	5,181	1,942	974
1902	3	17,031	10,599	4,678	4,227	1,366	1,725
1903	1	15,986	10,715	4,333	5,668	1,880	2,504
1904	83	19,235	12,594	4,223	5,120	2,231	2,935
1905	107	28,908	10 006	5,958	6,394	2,180	3,578

Year.	Copper and Brass Mnfs.			Cryolite.	Fluorspar.	Glass and Glassware.	Gold.	
	Fine.	Cartridge Cases, Coarse Wares, Etc.	Tombac, Scrap, Etc.				Bullion.	Specie.
1896	....	6,806	....	....	....	....	$23,091,119	$25,150,000
1897	....	5,409	....	....	....	....	21,472,940	7,150,000
1898	....	6,278	....	....	....	....	3,223,732	52,061,000
1899	....	5,846	....	....	....	....	3,223,068	30,548,500
1900	7,618	4,906	4,421	315	12,749	136,793	3,712,841	24,562,500
1901	6,904	4,968	5,013	367	13,436	129,720	5,755,668	6,848,000
1902	7,548	6,720	5,302	486	14,177	139,239	14,171,529	11,610,000
1903	8,134	7,540	5,525	349	13,028	155,921	15,273,353	6,854,750
1904	8,495	8,377	5,762	310	13,540	166,289	7,434,958	8,491,250
1905	9,519	10,708	6,263	286	15,019	165,000	3,884,901	13,996,541

Year.	Gold and Silver Manufactures. *Kg.*	Graphite.	Gypsum.	Iodine.	Infusorial Earth.	Iron and Steel.		
						Ore.	Pig.	Scrap.
1896	96,702	2,364	....	26	....	2,642,294	140,449	52,466
1897	101,429	2,422	....	26	....	3,230,391	90,885	38,102
1898	98,939	2,936	....	26	....	2,933,734	187,375	85,095
1899	107,021	2,703	....	26	....	3,119,878	182,091	53,103
1900	111,117	2,068	39,933	29	5,850	3,247,888	129,409	61,096
1901	98,564	1,667	40,397	27	5,776	2,389,870	150,448	153,399
1902	110,930	1,691	42,859	24	5,404	2,868,068	374,256	168,909
1903	112,633	1,810	51,874	29	5,770	3,343,510	418,072	109,245
1904	126,858	1,815	55,043	30	6,510	3,440,846	225,897	90,098
1905	134,185	1,971	52,886	27	6,741	3,668,563	380,824	118,485

Year.	Iron and Steel. (*Continued*).									
	Wrought, in Bars.	Wrought, in Sheets and Plates.	Castings.	Angles.	Blooms.	Rails.	Wire.	Manufactures.	Iron Oxide.	Ferrous Sulphate.
1896	259,461	(*g*)	153,652	178,887	49,529	129,413	207,116	309,882	.....	.....
1897	246,772	(*g*)	170,084	169,287	39,792	113,473	198,909	296,949	.....	.....
1898	263,698	(*g*)	193,039	204,705	34,964	123,839	188,713	312,948	.....	.....
1899	193,933	(*g*)	205,935	221,165	23,438	109,813	154,332	332,670	.....	.....
1900	172,533	167,363	196,759	215,641	33,627	155,656	169,839	207,208	1,052	3,829
1901	329,513	255,627	211,124	342,447	201,716	180,978	247,758	228,956	1,549	4,125
1902	361,216	273,021	247,404	382,238	636,427	366,815	233,513	242,244	1,755	4,360
1903	348,929	278,934	297,299	419,555	638,182	378,611	254,975	280,022	2,006	3,986
1904	298,621	256,186	288,611	373,238	395,990	211,049	267,429	295,741	2,093	3,514
1905	323,349	280,173	309,165	405,269	472,943	284,816	311,672	417,054	2,188	4,495

Year.	Lead.						Lime, Chloride of.	Magnesite.
	Ore.	Pig and Scrap.	Litharge.	White.	Red.	Manufactures.		
1896	.....	24,828	.....	16,350	.....	......	......	.....
1897	.....	24,075	.....	14,786	.....	......	......	.....
1898	.....	24,867	.....	16,473	.....	......	......	.....
1899	.....	24,491	.....	16,360	.....	......	......	.....
1900	1,309	18,825	3,577	15,126	6,603	14,594	25,954	2,392
1901	891	20,820	4,876	16,966	7,776	13,496	32,705	2,485
1902	2,024	23,100	4,072	19,070	8,372	13,764	29,694	2,955
1903	1,270	30,243	5,175	20,765	7,617	14,955	28,849	2,812
1904	1,312	23,169	5,410	16,638	7,544	14,969	30,078	1,917
1905	1,496	32,515	4,466	16,475	8,902	17,676	30,667	2,552

Year.	Magnesium Chloride.	Manganese Ore.	Mineral Pigments.	Nickel.	Ozokerite.	Petroleum Products.(*f*)		Phosphorus.
						Illuminating Oil.	Lubricating Oil.	
1896	......	7,178	......	143	.....	...	.....	...
1897	......	8,615	......	169	.....	...	.....	...
1898	......	4,810	......	203	.....	...	.....	...
1899	......	7,040	......	295	.....	...	.....	...
1900	13,375	2,454	13,958	268	1,592	843	1,455	170
1901	16,102	5,584	12,671	390	1,700	655	963	149
1902	14,757	4,528	14,392	689	1,856	824	1,177	260
1903	17,008	11,138	15,161	700	2,027	701	1,975	286
1904	16,706	5,536	16,395	1,203	2,447	760	1,763	236
1905	21,673	4,116	17,603	1,034	2,757	7,286	1,746	228

Year.	Phosphate Rock.	Potassium Salts.						Potassium and Potassium-Magnesium Sulphate.
		Bicarbonate.	Cyanide.	Chloride.	Hydroxide.	Iodide.	Nitrate.	
1896	5,548	12,673	55	85,862	......	142	11,323	......
1897	4,000	13,100	1,086	80,389	......	124	8,986	......
1898	5,100	13,456	1,907	96,236	......	135	10,969	......
1899	2,504	11,917	1,645	101,045	......	145	15,146	......
1900	1,123	15,761	1,338	114,469	15,379	138	14,744	38,125
1901	2,260	15,567	2,089	118,959	14,892	145	13,439	37,216
1902	1,103	14,041	3,257	106,925	13,804	152	9,734	40,487
1903	4,342	13,121	2,017	125,302	20,006	154	9,671	56,455
1904	3,222	10,777	3,290	140,765	24,963	174	10,405	64,400
1905	3,720	11,963	4,005	156,440	22,246	170	12,140	67,286

Year.	Pumice Stone.	Pyrites.	Quick-Silver.	Salt.	Silica, Sand, Marl, etc.	Silver.		Slag and Slag Wool.
						Bullion. *Kg.*	Specie. *Kg.*	
1896	...	16,833	(*e*)	214,060	267,173	305,826	31,648	17,214
1897	...	15,387	(*e*)	....	652,248	371,086	27,687	27,723
1898	...	19,220	97	225,548	910,354	348,733	46,932	29,931
1899	...	16,985	23	241,036	872,292	294,039	......	25,565
1900	561	24,936	23	236,291	822,840	284,853	31,799	32,494
1901	699	23,680	27	286,424	832,335	328,723	31,915	27,269
1902	691	35,370	109	328,324	713,568	372,390	24,049	22,726
1903	794	32,611	62	399,183	783,210	275,259	46,008	14,673
1904	943	30,666	43	347,351	980,673	282,017	43,986	38,587
1905	939	35,195	48	284,203	1,272,339	428,298	30,097	28,032

Year.	Slag. Basic.	Sodium Salts.						Sodium and Potassium Salts.	
		Bicarbonate.	Carbonate.	Hydroxide.	Nitrate. (Chile Saltpeter)	Soda, Calcined.	Sulphate and Sulphite.	Chromates.	Sulphides.
1896	134,257	.....	.....	.....	9,078	41,106	......	.....	.....
1897	169,336	.....	.....	.....	11,364	45,672	......	.....	.....
1898	187,598	.....	.....	.....	12,884	37,106	......	.....	.....
1899	199,382	.....	.....	.....	13,910	40,566	......	.....	.....
1900	174,563	1,314	1,392	1,913	14,159	44,316	41,572	3,741	2,461
1901	202,738	1,086	1,382	4,926	13,481	45,967	45,462	2,791	2,763
1902	162,062	954	2,449	5,650	14,737	33,109	56,748	2,656	4,565
1903	216,191	1,016	2,982	5,886	17,583	46,086	47,660	2,977	5,845
1904	258,767	1,524	3,050	5,084	21,075	43,590	45,506	2,272	5,489
1905	270,905	1,881	4,113	5,925	20,531	46,768	54,377	2,133	6,569

Year.	Stassfurt Salts.	Stone.						Strontium.	
		Crude or Simply Hewn.	Limestone.	Marble.	Grinding, Mill, Polishing, Whetstones, etc.	Slate.	Lithographic Stone.	Carbonate.	Salts.
1896	285,023	620,394	77,641	.....	......	5,796	.....	...	.....
1897	337,577	545,723	77,905	.....	......	4,948	.....	...	.....
1898	370,829	556,561	84,892	.....	......	4,434	.....	...	.....
1899	367,828	597,286	84,915	.....	......	3,034	.....	...	.....
1900	468,277	675,605	78,758	3,248	22,881	3,123	5,409	74	496
1901	592,347	600,654	76,079	3,290	21,469	3,076	5,214	384	1,022
1902	499,220	597,136	81,078	2,936	20,536	3,257	5,264	762	1,546
1903	501,385	594,459	82,774	3,079	20,805	2,858	5,679	819	1,389
1904	631,762	615,165	91,599	2,270	21,641	2,810	5,305	613	1,207
1905	852,454	765,008	91,753	1,407	22,723	2,538	5,862	613	1,386

Year.	Sulphur.	Sulphuric Acid.	Super-Phosphate	Tin.		Zinc.					Zinc White, Zinc-Gray, and Lithophon.
				Crude.	Mfrs.	Ore.	Spelter and Scrap.	Drawn or Rolled.	Mfrs.	Sulphate	
1896	.....	......	......	868	.....	37,959	58,082	16,227	.....	...	16,969
1897	.....	......	......	861	.....	30,047	51,341	17,453	.....	...	17,631
1898	.....	......	......	874	.....	30,408	51,324	14,477	.....	...	18,674
1899	.....	......	......	1,121	.....	25,192	46,334	18,281	.....	...	19,489
1900	1,146	37,738	77,118	1,626	1,568	34,941	51,899	16,709	2,346	382	20,729
1901	621	42,853	79,190	1,683	1,701	41,002	54,490	16,517	2,003	324	24,201
1902	576	47,666	77,818	2,271	1,952	46,965	70,292	17,015	2,251	330	28,400
1903	1,052	50,109	99,672	2,581	1,890	40,458	67,057	15,715	2,411	264	27,527
1904	1,418	52,696	129,925	2,965	2,122	40,488	70,063	17,917	3,163	332	26,898
1905	1,198	48,701	115,886	3,259	2,065	38,972	67,675	18,982	3,140	296	27,877

(*a*) From *Statistisches Jahrbuch für das Deutsche Reich.* (*b*) Includes witherite. (*c*) Represents unglazed tile and brick only. (*d*) Includes precipitated chalk. (*e*) Not reported. (*f*) Of domestic production only. (*g*) Included under manufactures.

GREECE.

The statistics of mineral production in Greece are summarized in the following tables:

MINERAL PRODUCTION OF GREECE. (a)
(In metric tons or dollars; 1 drachma=20 cents.)

Year.	Chrome Ore.	Emery.	Gypsum.	Iron Ore.	Iron Ore. Manganiferous.	Lead. Soft.	Lead Ore. Argentiferous.	Lead. Argentiferous.	Lead. Fume.	Lignite.
1895....	2,740	3,055	113	150,210	152,123	7	1,580	19,838	1,406	17,748
1896....	1,600	3,650	120	225,600	166,850	480	3,200	14,700	1,550	14,000
1897....	563	3,024	51	260,828	182,850	520	2,815	15,946	2,785	20,018
1898....	1,367	3,932	83	287,100	213,938	305	(b)	18,888	2,655	17,310
1899....	4,386	4,360	81	331,030	294,320	291	(b)	18,768	2,584	12,150
1900....	5,600	6,328	129	279,880	243,920	245	878	16,150	2,045	12,940
1901....	4,580	5,691	671	278,640	196,152	(b)	(b)	17,644	5,292	9,726
1902....	11,680	4,727	10	364,340	170,040	(b)	430	14,048	1,647	6,500
1903....	8,478	5,586	94	531,804	152,740	(b)	(b)	12,361	(b)	8,687
1904....	15,430	6,182	393	413,688	(c)239,635	(b)	(b)	12,590	(b)	10,000

Year.	Magnesite.			Manganese Ore.	Millstones. *Number.*	Puzzolan.	Sea Salt.	Sulphur.	Zinc Ore.	
	Crude.	Bricks.	Calcined.						Blende.	Calamine, Calcined.
1895....	11,096	890	1,402	7,250	15,325	29,810	22,238	1,480	2,710	21,321
1896....	11,600	892	1,514	15,500	6,757	31,300	22,800	1,540	1,750	20,950
1897....	11,311	826	686	11,868	6,975	42,600	20,421	358	3,118	22,817
1898....	14,829	516	129	14,097	18,500	70,700	25,250	135	1,139	30,906
1899....	17,184	542	3,087	17,600	12,563	46,375	37,125	1,150	1,137	21,770
1900....	17,277	534	807	8,050	13,386	49,426	22,411	891	(b)	18,751
1901....	13,410	500	2,009	14,166	16,400	80,223	23,079	3,212	454	17,764
1902....	27,103	935	4,730	14,960	13,564	45,400	25,200	1,391	(b)	18,670
1903....	25,657	(b)	(b)	9,340	11,000	(c)11,728	26,000	1,266	(c) 1,122	12,350
1904....	35,989	(b)	(b)	7,355	12,744	(c)18,888	27,000	569	(c)13,234	15,446

(a) Statistics up to 1903 communicated by E. Grohmann, Seriphos. (b) Not reported. (c) Exports.

INDIA.

The official statistics of mineral production in British India are summarized in the subjoined tables:

MINERAL PRODUCTION OF INDIA. (*a*)

(In metric tons or dollars: £1 = $5.)

Year.	Amber.	Coal.	Corundum.	Gold. (*c*)	Graphite.	Iron Ore.	Jade. (*e*)	Magnesite.
1895.....	(*b*)	4,441,681	(*b*)	$5,144,282	(*b*)	44,009	229	(*b*)
1896.....	(*b*)	3,909,764	(*b*)	7,085,432	(*b*)	50,559	215	(*b*)
1897.....	(*b*)	4,128,330	(*b*)	8,041,055	(*b*)	61,697	219	(*b*)
1898.....	$5,080	4,681,927	133	7,798,709	(*b*)	(*d*)42,524	196	(*b*)
1899.....	755	5,174,752	40	8,357,087	1,548	(*d*)52,832	228	(*b*)
1900.....	515	6,222,591	63	9,205,518	1,859	(*d*)57,912	142	(*b*)
1901.....	55	6,741,899	74	9,394,723	2,530	(*d*)58,725	206	(*b*)
1902.....	2,160	7,543,272	26	9,611,985	4,648	(*d*)77,273	137	3,597
1903.....	2,070	7,557,400	(*b*)	11,203,926	3,448	(*d*)62,337	191	838
1904.....	4,190	7,456,176	(*b*)	11,513,340	2,955	64,980	171	1,193

Year.	Manganese Ore.	Mica. (*e*)	Petroleum. *Gallons.*	Rubies.	Salt.	Saltpeter (Potassium nitrate.)	Tin Ore.
1895..........	16,070	261	13,013,990	$72,839	1,120,780	17,931	22
1896..........	57,782	452	15,057,094	171,884	1,043,171	21,425	82
1897..........	74,862	652	19,128,828	200,613	937,932	26,845	62
1898..........	61,419	527	22,234,438	289,750	1,043,862	21,224	40
1899..........	88,524	497	32,934,007	454,240	977,269	18,555	64
1900..........	129,865	1,025	37,729,211	486,630	1,071,877	20,189	94
1901..........	122,831	1,505	50,075,117	522,380	1,208,933	17,711	63
1902..........	160,311	739	56,607,688	434,475	1,116,797	18,005	91
1903..........	174,563	926	87,859,069	444,095	1,021,581	20,861	100
1904..........	136,385	888	118,491,382	453,060	1,319,535	14,318	63

(*a*) (*b*) Not reported. (*c*) £1 = $4.866. (*d*) Production of iron ore in Bengal only. (*e*) Production and exports the same as for fiscal year ending March 31.

ITALY

The following tables itemize the statistics of the production and the foreign commerce of mineral and metallurgical products in Italy:

MINERAL PRODUCTION AND REFINED PRODUCTS OF ITALY. (a)

(In metric tons or dollars; 5 lire=$1.)

Year.	Alum.	Aluminum Sulphate.	Alunite.	Antimony.	Antimony Ore.	Asphalt, Mastic and Bitumen.	Asphaltic Rock.	Barytes.
1895.......	995	2,950	7,000	423	2,241	14,491	46,713	(b)
1896.......	850	2,390	6,000	538	5,086	12,490	45,456	(b)
1897.......	1,030	2,310	6,500	404	2,150	18,644	55,339	(b)
1898.......	1,165	2,915	7,000	380	1,931	17,813	93,750	12,400
1899.......	945	2,330	5,800	581	3,791	41,732	81,987	12,545
1900.......	1,097	2,403	5,200	1,174	7,609	33,127	101,738	14,003
1901.......	1,075	2,260	4,900	1,721	8,818	31,814	104,111	13,245
1902.......	....	....	8,200	....	6,116	....	64,245	....
1903.......	....	....	8,100	905	6,927	35,757	89,078	....
1904.......	2,490	2,210	8,000	836	5,712	30,817	111,390	250

Year.	Borax. Refined.	Boric Acid.		Coal. (c)	Coal. (Briquettes).	Coke.	Copper.	
		Crude.	Refined.				Ore.	Ingot, etc.
1895.....	944	2,633	253	305,321	451,470	394,043	83,670	2,375
1896.....	943	2,616	253	276,197	422,409	426,906	90,408	2,842
1897.....	990	2,704	260	314,222	549,050	430,617	93,377	2,980
1898.....	702	2,650	166	341,327	594,500	469,228	95,128	3,230
1899.....	709	2,674	129	388,534	566,000	485,951	94,764	3,032
1900.....	858	2,491	283	479,896	703,740	487,831	95,644	2,797
1901.....	544	2,558	347	425,614	754,800	490,803	107,750	3,097
1902.....	....	2,763	....	414,569	....	....	101,142	....
1903.....	....	2,583	....	346,887	724,993	554,559	114,823	....
1904.....	569	2,624	314	362,151	903,610	607,297	157,503	2,313

Year.	Gold.		Graphite.	Iron and Steel.				
	Ore.	Bullion.		Ore.	Pig.	Bar, Sheet, Pipe, Wire, etc.	Tin Plate.	Steel.
1895.....	7,099	$186,074	2,657	183,371	9,213	163,824	5,860	50,314
1896.....	7,659	172,552	3,148	203,966	6,987	139,991	2,918	65,955
1897.....	10,723	209,998	5,650	200,709	8,393	149,944	6,500	63,940
1898.....	9,549	124,869	6,435	190,110	12,387	167,499	7,200	87,467
1899.....	11,859	75,294	9,990	236,549	19,218	197,730	8,000	108,501
1900.....	5,840	38,212	9,720	247,278	23,990	190,518	10,000	115,887
1901.....	890	2,725	10,313	232,299	15,819	180,729	7,550	123,310
1902.....	1,215	....	9,210	240,705	....	....	....	....
1903.....	5,734	41,933	7,920	374,790	90,744	177,392	11,275	154,134
1904.....	6,746	43,063	9,765	409,460	112,598	181,385	16,655	177,086

Year.	Lead. Ore.	Lead. Pig.	Manganese Ore.	Manganiferous Iron Ore.	Marble.	Petroleum, Crude.	Petroleum, Benzine, etc.	Pumice Stone.
1895.....	30,632	20,353	1,569	5,860	186,900	3,594	4,191	(b)
1896.....	33,545	20,786	1,890	10,000	209,428	2,524	2,734	(b)
1897.....	36,200	22,407	1,634	21,262	236,958	1,932	3,392	(b)
1898.....	33,930	24,543	3,002	11,150	271,725	2,015	5,040	2,766
1899.....	31,046	20,543	4,356	29,874	313,744	2,242	5,384	7,300
1900.....	35,103	23,763	6,014	26,800	310,336	1,683	6,077	7,000
1901.....	43,449	25,796	2,181	24,290	334,146	2,246	4,211	8,300
1902.....	42,330	....	2,477	23,113	....	2,633	....	....
1903.....	42,443	22,126	1,930	4,735	....	2,486	4,577	....
1904.....	42,846	23,475	2,836	*Nil.*	390,118	3,543	6,388	11,600

Year.	Pyrites. (Cupriferous in part).	Quicksilver. Ore.	Quicksilver. Metal.	Salt. Brine.	Salt. Rock.	Salt. Sea.	Silver. Ore.	Silver. Bullion. *Kg.*
1895............	38,586	10,504	199	10,605	18,710	448,335	870	44,189
1896............	45,728	14,305	186	11,974	17,300	422,555	640	38,075
1897............	58,320	20,659	192	11,725	19,801	429,253	405	45,313
1898............	67,191	19,201	173	11,546	18,199	451,426	435	43,437
1899............	76,538	29,322	205	11,021	18,721	363,826	540	33,645
1900............	71,616	33,930	260	10,890	18,331	338,034	584	31,169
1901............	89,376	38,614	278	10,690	23,054	401,443	511	32,464
1902............	93,177	44,261	....	10,581	23,677	424,239	421	....
1903............	101,455	55,528	312	10,962	25,911	451,633	405	24,388
1904............	112,004	60,403	352	11,878	18,638	433,810	143	24,943

Year.	Sulphur. Crude (Fused).	Sulphur. Ground.	Sulphur. Refined.	Talc. Ground.	Zinc. Ore.	Zinc. Spelter.
1895........................	370,766	91,517	75,329	(b)	121,197	*Nil.*
1896........................	426,353	89,292	71,072	(b)	118,171	*Nil.*
1897........................	496,658	69,178	85,872	(b)	122,214	250
1898........................	502,351	146,001	99,494	12,760	132,099	250
1899........................	563,697	161,509	110,213	11,000	150,629	251
1900........................	544,119	167,466	157,957	14,415	139,679	547
1901........................	563,096	171,252	141,431	11,770	135,784	511
1902........................	....	....	....	....	131,965	....
1903........................	553,751	139,376	139,464	6,300	157,521	126
1904........................	527,563	189,266	163,695	6,740	148,365	189

(*a*) From *Rivista del Servizio Minerario.* (*b*) Not reported. (*c*) Includes anthracite, lignite, fossil wood and bituminous schist.

MINERAL IMPORTS OF ITALY. (*a*)

(In metric tons or dollars; 5 lire = $1.)

Year.	Antimony	Arsenic. *Kg.*	Asbestos.	Asphaltum.	Barytes.	Borax and Boric acid.	Cement and Hydraulic Lime.	Chalk.
1896.................	38	(b)	851	11,892	549	166	12,810	15,716
1897.................	66	2,604	619	1,632	578	253	16,680	28,937
1898.................	58	700	1,186	1,150	860	147	12,029	18,252
1899.................	64	600	1,675	1,473	936	123	14,391	13,738
1900.................	37	900	1,645	1,933	859	122	15,494	18,436
1901.................	49	1,800	2,019	1,450	825	232	14,872	20,731
1902.................	80	1,200	1,536	1,020	1,170	516	13,732	15,216
1903.................	98	4,400	1,691	1,567	1,099	504	15,047	10,063
1904.................	131	3,700	2,174	2,604	1,875	271	15,260	6,891
1905.................	117	3,400	1,806	3,252	1,444	112	15,797	5,556

Year.	Clay Products. Brick, Tile, etc.	Clay Products. Kaolin.	Clay Products. Terra cotta.	Coal.	Copper Ore.	Copper Cement.
1896.................................	18,504	3,775	2,675	4,081,218	484	1,150
1897.................................	19,086	5,719	2,167	4,259,643	1,611	1,049
1898.................................	21,681	9,079	2,122	4,431,524	5,471	2,040
1899.................................	22,410	12,105	2,200	4,859,556	2,777	1,328
1900.................................	25,702	9,595	2,031	4,947,180	5,290	1,298
1901.................................	35,534	12,809	2,482	4,838,994	11,047	1,987
1902.................................	29,344	14,165	2,537	5,406,069	9,422	2,299
1903.................................	....	11,033	2,883	5,546,823	9,459	649
1904.................................	....	18,610	3,236	5,904,578	8,104	309
1905.................................	....	15,315	4,080	6,437,539	6,879	486

Year.	Copper, Brass and Bronze.	Copper and Iron Sulphates.	Glass and Manufactures.	Gold.			Graphite.	Iron.		
				Specie. *Kg.*	Unrefined. *Kg.*	Manufactures: *Kg.*		Ore.	Pig.	Wrought.
1896....	6,955	24,255	9,322	1,004	2,517	1,515	204	594	119,491	4,820
1897....	7,999	28,878	11,182	444	807	1,375	315	5,831	156,019	3,801
1898....	7,433	25,560	10,399	154	507	1,844	382	8,723	169,059	4,076
1899....	7,334	27,408	10,303	181	326	1,390	608	20,799	191,613	4,158
1900....	9,249	32,127	11,356	188	309	1,348	982	19,205	160,686	7,405
1901....	8,659	32,058	10,448	1,115	494	1,547	102	4,054	159,972	5,695
1902....	10,865	25,107	11,813	8,967	479	1,269	60	4,314	155,143	6,603
1903....	9,588	24,566	12,830	44,218	1,396	1,220	63	5,937	126,756	6,380
1904....	15,198	37,298	13,957	8,364	1,961	1,640	52	4,390	149,130	6,740
1905....	18,188	30,684	14,227	46,509	5,768	1,799	107	4,745	136,077	7,616

Year.	Iron and Steel.		Lead.			Lead. Oxide and Carbonate	Mineral Paints.	Nickel Alloys and Manufactures.	Petroleum.
	Plates, Rods and Manufactures.	Scrap.	Ore. (*c*)	Metal and Alloys in Pigs.	Mnfrs.				
1896......	52,202	162,035	9,730	1,166	192	523	852	411	70,217
1897......	88,895	130,938	14,854	1,178	247	580	888	432	68,973
1898......	98,564	138,426	10,947	1,431	435	647	692	258	70,654
1899......	142,645	245,616	7,476	3,990	249	662	958	250	71,391
1900......	149,263	197,415	9,134	3,248	233	557	958	232	73,089
1901......	146,970	148,305	9,063	2,926	268	815	865	476	69,298
1902......	136,735	198,914	1,680	7,563	288	846	670	561	68,781
1903......	160,705	206,036	689	5,398	273	768	859	525	68,220
1904......	160,389	246,359	2,187	4,541	247	871	940	652	69,233
1905......	163,358	276,326	465	6,764	295	686	974	574	66,493

Year.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Potassium Sulphate.	Precious Stones. Manufactures.	Quick-Silver.	Silver. *Kg.*			Slag.
						Specie.	Unrefined in Bars.	Manufactures.	
1896........	(*b*)	9,841	431	$1,505,464	30	22,751	2,291	6,533	30,275
1897........	(*b*)	11,012	562	2,033,714	30	26,008	2,434	5,286	37,201
1898........	65,126	11,047	928	1,629,008	39	8,241	991	5,673	51,199
1899........	116,283	12,370	1,297	1,323,317	62	20,605	1,782	4,881	56,549
1900........	140,281	14,077	1,670	1,801,264	49	29,291	2,678	4,358	32,254
1901........	142,108	14,693	1,411	2,967,041	36	35,089	4,391	4,213	7,312
1902........	159,341	17,617	1,566	3,885,484	57	28,662	8,768	3,455	5,634
1903........	172,328	17,528	1,353	4,866,327	28	81,373	12,541	5,893	8,849
1904........	217,162	14,846	1,663	4,896,867	25	67,520	15,885	4,409	3,821
1905........	240,144	17,752	1,804	6,563,084	57	51,997	20,697	5,437	5,326

Year.	Sodium Salts.			Tin.		Zinc.			
	Carbonate.	Nitrate (Crude).	Sod. and Pot. Nitrates, Refined.	Block.	Mnfrs.	Ore.	Oxide.	Spelter and Old.	Mnfrs.
1896.........	18,927	11,685	541	1,763	91	(*b*)	540	2,596	3,482
1897.........	20,721	16,400	917	1,520	81	(*b*)	570	3,278	3,556
1898.........	20,845	19,961	702	1,722	109	216	573	2,813	3,200
1899.........	22,654	22,385	671	1,240	96	(*b*)	804	3,498	3,221
1900.........	23,215	27,706	511	1,643	56	85	1,034	3,627	3,543
1901.........	21,956	40,498	315	1,858	91	23	813	3,991	4,079
1902.........	26,133	24,483	314	2,114	110	131	904	3,805	4,167
1903.........	24,753	43,480	638	2,288	130	46	1,416	4,541	4,461
1904.........	27,747	32,283	613	2,170	150	362	1,124	5,202	4,168
1905.........	29,066	46,517	689	2,304	103	14	1,246	5,997	4,701

MINERAL EXPORTS OF ITALY. (*a*)

(In metric tons or dollars; 5 Lire=$1.)

Year.	Antimony.	Asbestos.	Asphaltum.	Barytes.	Borax and Boric Acid.	Cement and Hydraulic Lime.	Chalk.
1896..............................	361	130	13,729	66	2,719	3,871	5,593
1897..............................	271	170	15,310	143	1,618	5,330	7,556
1898..............................	338	208	19,465	70	2,167	5,192	6,744
1899..............................	240	245	26,402	45	2,872	5,462	5,386
1900..............................	467	261	24,287	40	2,114	6,860	2,980
1901..............................	765	302	21,856	32	2,190	8,463	3,428
1902..............................	359	144	20,884	91	1,847	7,930	4,215
1903..............................	314	222	24,303	35	901	6,325	3,802
1904..............................	107	163	14,880	70	1,122	7,810	4,089
1905..............................	132	236	23,740	162	2,255	8,445	5,007

Year.	Clay Products.				Coal.	Copper Ore.	Brass and Bronze.	Copper, and Iron Sulphate.
	Brick, Tile, etc.	Kaolin.	Porcelain.	Terra-Cotta.				
1896	143,648	49	99	2,852	18,924	3,603	643	71
1897	125,925	93	138	2,472	23,191	2,408	641	18
1898	125,614	94	86	2,751	17,749	2,356	857	25
1899	136,402	94	124	2,796	20,803	1,148	1,734	20
1900	122,388	179	96	3,051	23,926	1,179	1,209	60
1901	108,057	368	266	3,325	25,594	9	659	20
1902	133,922	*Nil*	154	3,378	33,374	11	874	39
1903	....	133	192	4,482	29,219	15	952	44
1904	....	18	242	4,581	35,149	43	939	29
1905	....	86	229	5,194	38,555	77	812	260

Year.	Glass and Manufactures.	Gold.		Graphite.	Iron.			Iron and Steel Plates, Rods and Manufactures.
		Specie.	Unrefined. *Kg.*		Ore.	Pig.	Wrought.	
1896	5,867	$1,539,460	2,517	3,727	187,059	1,378	427	1,819
1897	6,022	996,960	1,381	4,164	207,619	498	1,434	7,148
1898	5,209	1,590,920	1,739	5,145	217,556	840	699	4,606
1899	5,930	1,281,540	1,162	8,114	234,515	378	611	4,817
1900	6,158	1,453,900	2,763	7,820	170,286	329	440	5,610
1901	8,182	1,159,400	2,955	7,169	121,592	311	499	5,120
1902	5,131	1,176,140	733	7,098	209,070	395	1,054	5,540
1903	5,344	507,160	1,291	7,068	98,319	810	1,670	6,594
1904	5,490	1,011,840	1,494	7,433	2,577	229	847	9,945
1905	5,019	655,340	1,731	6,811	11,358	1,395	2,346	12,228

Year.	Lead.				Mineral Paints.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Quicksilver.	Salt.
	Ore.	and Lead Alloys in Pigs.	Manufactures.	Oxide and Carbonate.					
1896	4,731	1,419	1,441	489	2,412	(*b*)	88	155	171,740
1897	4,747	2,790	1,410	461	2,318	(*b*)	66	236	176,520
1898	4,492	5,870	1,764	414	2,884	(*b*)	85	244	126,860
1899	3,129	2,497	910	389	2,784	(*b*)	120	223	114,050
1900	3,741	5,018	1,408	367	2,977	1,726	142	259	112,900
1901	3,977	4,463	2,128	410	2,913	1,290	198	301	114,210
1902	3,354	5,650	2,258	404	2,953	894	136	215	145,190
1903	5,041	2,911	1,934	426	3,305	2,942	233	222	144,910
1904	5,524	1,954	1,887	347	3,231	2,812	162	266	130,940
1905	4,311	976	2,043	310	3,632	3,509	238	243	116,040

Year.	Silver. *Kg.*		Slag.	Sodium Salts.			Stone.			
	Specie.	Unrefined.		Carbonate.	Nitrate. (Crude.)	Sod. and Pot. Nitrates, Refined.	Alabaster. (Crude.)	Building Stone.	Marble. (Crude.)	Marble and Alabaster.
1896	28,377	26,854	4,753	279	51	306	289	23,580	80,750	68,639
1897	72,605	50,503	8,847	275	151	344	269	36,229	83,081	62,750
1898	8,241	68,607	6,861	391	79	256	457	35,945	88,404	68,150
1899	32,085	32,432	4,898	438	136	124	714	53,904	98,485	84,104
1900	10,501	25,310	4,222	486	58	129	489	54,051	61,650	72,619
1901	14,446	42,325	3,261	377	116	59	474	53,668	96,631	73,599
1902	10,978	20,427	3,615	446	346	259	727	74,036	112,967	83,172
1903	4,377	9,486	4,929	482	781	492	605	69,473	130,316	87,079
1904	3,834	24,165	4,458	376	363	230	800	80,337	131,087	82,911
1905	2,371	25,947	9,725	214	424	159	935	116,110	132,765	94,295

Year.	Sulphur.	Tin.		Zinc.			
		Block.	Manufactures.	Ore.	Oxide.	Spelter and Scrap.	Manufactures.
1896	356,370	10	89	115,454	48	33	8
1897	358,932	29	109	133,125	189	309	63
1898	405,823	34	177	130,064	110	156	14
1899	424,018	69	176	140,107	123	227	21
1900	479,139	147	153	111,870	102	359	24
1901	414,018	202	187	103,020	140	349	18
1902	439,242	236	174	114,894	122	338	66
1903	461,289	173	180	116,449	116	591	51
1904	437,067	171	151	126,393	483	263	46
1905	381,128	285	107	117,810	173	434	48

(*a*) From *Statistica del Commercio speciale di Importazione e di Esportazione*. (*b*) Not reported. (*c*) Includes argentiferous lead ore.

JAPAN.

The total mineral production of the Japanese Empire, according to the latest available returns, is shown in the following table, in metric tons, unless otherwise specified: (*a*)

Year.	Antimony.		Arsenic. *Kg.*	Coal.	Copper.	Gold. *Kg.*	Graphite.	Iron. Pig.	Lead.
	Sulphate.	Metal.							
1894	1,172	403	5,387	4,300,370	19,814	806	1,091	19,474	1,577
1895	1,061	641	7,343	4,770,313	19,103	935	77	25,863	1,978
1896	828	517	6,043	5,100,005	20,114	964	215	27,420	1,958
1897	358	823	13,039	5,147,103	20,425	1,037	391	28,040	772
1898	1,005	233	7,129	6,643,047	21,023	1,159	347	23,611	1,703
1899	712	229	5,077	6,668,608	19,421	1,675	53	23,066	1,963
1900	809	349	4,669	7,370,667	24,317	2,124	94	24,841	1,878
1901	118	429	10,312	8,884,812	27,392	2,475	88	29,449	1,803
1902	88	528	12,188	9,588,910	29,034	2,975	97	32,130	1,644
1903	153	434	6,000	10,088,845	33,245	3,140	114	33,870	1,728

Year.	Manganese Ore.	Petroleum. *Gallons.*	Phosphates.	Pyrite.	Quicksilver. *Kg.*	Salt. *Hectoliters.*	Silver. *Kg.*	Sulphur.	Tin.
1894	13,368	(*c*)5,426,071,	(*b*)	(*b*)	1,547	11,411,275	79,222	18,787	38.7
1895	17,142	7,118,962	(*b*)	(*b*)	481	(*b*)	74,815	15,557	48.3
1896	17,967	(*c*)7,440,206	(*b*)	(*b*)	1,762	(*b*)	64,303	12,540	50.0
1897	15,448	9,179,474	(*b*)	7,626	2,678	(*b*)	54,289	13,582	47.6
1898	11,497	11,145,457	(*b*)	8,726	1,399	11,482,422	60,436	10,321	42.7
1899	11,336	18,844,034	(*b*)	8,376	....	10,483,082	56,161	10,237	18.5
1900	15,831	30,470,068	(*b*)	16,166	270	11,890,361	58,799	14,439	12.3
1901	16,270	39,056,820	(*b*)	17,589	750	12,463,771	54,739	16,548	14.1
1902	10,844	34,850,129	196	18,580	1,418	11,042,192	57,635	18,287	18.6
1903	5,616	50,724,174	191	16,149	206	6,574,890	58,704	22,914	19.0

(*a*) From *Résumé Statistique de l'Empire du Japon*, Tokio. (*b*) Not reported. (*c*) Crude petroleum.

MEXICO.

Owing to the delay in the appearance of Mexican statistics of production, we are unable to reproduce them here. Exports may, however, be taken as indicating the condition of the mining industry.

MINERAL EXPORTS OF MEXICO. (*a*)
(In metric tons or Mexican dollars.)

Year.	Antimony Ore.	Coal.	Copper.		Gold.				
			Ore.	Ingot.	Ore.	Bullion.	Specie.	Cyanide.	Sulphide.
1895	600	61,686	3,006	20,429	$103,773	$4,920,504	$175,098	$31,231	$3,026
1896	3,231	75,541	144	20,659	206,874	5,533,789	261,078	161,784	44,890
1897	5,873	105,298	1,094	16,858	365,226	6,220,765	202,223	226,986	33,916
1898	5,932	118,553	13,146	10,362	1,037,202	6,493,735	(*b*)	294,730	64,061
1899	10,382	113,192	223	25,293	335,849	7,017,286	183,474	115,961	266,782
1900	2,313	38,676	408	27,970	306,392	7,435,864	192,456	128,675	177,193
1901	5,103	17,281	5,576	33,818	284,722	8,324,681	210,431	178,803	81,744
1902	1,280	3,406	6,101	63,609	303,979	9,079,371	129,899	78,295	40,658
1903	7,302	1,840	10,912	51,716	264,503	9,693,692	54,636	85,465	124,020
1904	....	....	....	....	....	....	....	....	....

Year.	Graphite.	Gypsum.	Lead.		Silver.					
			Ore.	Base Bullion.	Ore.	Bullion.	Specie.	Sulphide.	Cyanide.	Slag.
1895.	794	1,340	568	50,122	$10,977,079	$22,178,294	$18,300,553	$555,475	$14,649	$72,590
1896.	795	2,050	167	48,663	9,971,053	28,565,843	18,737,331	1,495,306	38,049	64,121
1897.	759	2,095	2	60,029	11,401,176	35,775,125	21,925,347	1,663,581	123,246	39,800
1898.	(*b*)	1,650	(*b*)	60,918	11,048,358	37,137,599	(*b*)	1,663,501	257,342	46,488
1899.	2,305	1,050	1	67,441	10,766,099	37,585,911	5,580,834	1,929,085	76,942	4,810
1900.	2,561	1,600	468	74,944	12,495,524	41,468,745	22,679,655	1,893,646	67,607	87,883
1901.	762	800	(*b*)	79,097	9,615,939	36,348,374	12,038,158	2,141,685	259,282	93,549
1902.	1,434	....	118	107,366	4,108,088	45,796,576	17,753,526	1,978,919	108,344	132,093
1903.	1,404	....	11	100,532	11,781,048	48,276,797	16,167,673	1,642,627	135,561	289,900
1904.	......	....	......	....	....	....	....	....	....	....

(*a*) From the *Estadistica Fiscal*. The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. (*b*) Not reported.

NORWAY.

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (*a*)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Apatite (*b*)	Chrome Ore.	Copper.		Feldspar.	Gold.	Iron.		
			Ore.	Ingot.			Ore.	Pig and Cast.	Bars and Steel.
1895......	1,601	....	21,896	958	9,780	$2,160	1,250	348	379
1896......	1,106	....	29,910	1,067	12,223	9,450	2,000	335	400
1897......	872	....	27,606	1,064	17,392	675	3,627	417	452
1898......	3,593	....	37,047	941	11,355	1,539	4,425	231	379
1899	1,500	41	43,358	1,209	19,260	2,700	4,576	406	666
1900......	300	165	46,858	1,280	17,609	2,430	17,925	444	614
1901......	738	85	40,726	1,073	18,323	2,700	42,252	261	*Nil*
1902......	2,295	22	40,499	1,347	19,591	36,990	53,675	527	*Nil*
1903......	1,795	*Nil*	35,417	1,382	18,590	83,700	53,475	509	*Nil*
1904......	1,456	154	36,891	(*c*)	20,835	*Nil*	45,328	(*c*)	(*c*)

Year.	Nickel.		Pyrites, Iron and Copper.	Rutile.	Silver.		Zinc Ore
	Ore.	Metal.			Ore and Native Silver.	Metal. *Kg.*	
1895..................	494	17	49,005	28	490	5,000	(*c*)
1896..................	*Nil*	16	60,507	30	527	4,664	450
1897..................	*Nil*	*Nil*	94,484	32	642	5,372	908
1898..................	*Nil*	*Nil*	89,763	35	497	4,802	320
1899..................	220	5	95,636	30	429	4,600	379
1900..................	1,888	13	98,945	40	475	4,600	204
1901..................	2,018	40	101,894	55	519	5,680	90
1902..................	4,040	60	121,247	*Nil*	471	6,220	30
1903..................	5,670	75	129,939	25	480	7,269	335
1904..................	5,352	(*c*)	133,603	25	(*c*)	8,064	42

(*a*) *Tabeller vedkommende Norges Bergværksdrift, Statistik Aarbog for Kongeriget Norge.* (*b*) Exports which represent production. (*c*) Not reported.

MINERAL IMPORTS OF NORWAY.
(In metric tons.)

Year.	Borax. *Kg.*	Cement and Hydraulic Lime.	Coal.	Copper and Brass.		Glass and Glassware.
				Plates and Bars.	Wares.	
1895..............................	40,154	14,555	1,120,976	1,262	417	3,343
1896..............................	38,305	16,028	1,136,087	1,074	479	3,729
1897..............................	44,495	18,734	1,229,966	1,140	591	4,262
1898..............................	71,590	25,403	1,232,792	1,064	807	3,905
1899..............................	62,060	33,652	1,478,080	1,000	1,120	3,229
1900..............................	71,124	24,511	1,520,162	696	1,164	2,874
1901..............................	68,000	20,993	1,413,228	1,018	761	1,793
1902..............................	(c)	18,984	1,547,089	1,118	(c)	(c)
1903..............................	(c)	17,906	1,528,612	899	309	690
1904..............................	54,953	12,845	1,394,343	688	866	1,158

Year.	Iron and Steel.						Bars, Rods, etc.	Other Mnfrs.
	Pig and Cast.	Bars, Hoops, etc. Wrought Iron.	Anchors, Cables and Chains.	Rails.	Nails and Spikes.	Steel.		
1895.......	27,735	24,985	1,152	10,337	1,796	3,654	17,035	6,344
1896.......	29,764	26,552	1,090	4,315	1,760	2,754	17,930	6,831
1897.......	35,055	29,038	1,367	7,637	2,097	4,350	23,350	10,695
1898.......	41,227	26,203	1,485	10,327	2,087	2,428	26,894	17,182
1899.......	46,680	25,379	1,394	8,137	1,529	2,652	32,192	21,400
1900.......	32,113	23,010	1,203	11,952	1,219	2,085	29,318	17,493
1901.......	31,660	20,672	1,708	22,959	1,808	1,905	31,184	18,372
1902.......	30,590	26,685	2,103	15,316	2,205	1,754	36,288	22,069
1903.......	31,869	21,977	1,807	4,631	1,261	1,958	42,098	18,855
1904.......	30,975	24,094	2,109	5,814	928	1,610	42,013	5,462

Year.	Lead in Pigs and Sheets.	Lead, White and Zinc Oxide.	Petroleum and Paraffin.	Potash.	Salt.	Saltpeter.	Soda.	Sulphur. (b)	Tin in Blocks, etc.	Zinc in Bars, Plates, etc.
1895...	657	1,068	27,978	562	123,933	227	5,234	7,271	95	978
1896...	653	1,192	35,823	945	117,920	308	5,156	9,347	142	1,101
1897...	848	1,119	39,810	919	164,572	277	5,492	10,701	236	1,102
1898...	732	1,491	36,504	754	127,341	477	4,823	9,589	257	1,370
1899...	869	1,296	42,182	802	134,583	278	4,555	10,734	546	1,509
1900...	670	1,216	39,657	638	143,365	356	4,576	14,827	149	1,254
1901...	590	1,321	47,011	518	127,607	208	5,220	11,149	141	1,027
1902...	(c)	(c)	(c)	(c)	141,415	315	(c)	(c)	(c)	1,104
1903...	311	(c)	58,822	457	143,110	245	4,200	8,829	106	1,015
1904...	607	(c)	50,543	477	153,699	321	3,197	12,181	176	940

MINERAL EXPORTS OF NORWAY. (a)

(In metric tons.)

Year.	Apatite.	Clay Products.		Copper.			Feldspar.
		Bricks. *Thousands.*	Earthenware.	Ore.	Ingot.	Scrap.	
1895	1,600	5,016	286	20,283	1,515	810	9,780
1896	1,160	10,008	365	30,367	1,276	712	12,223
1897	872	11,711	260	15,111	1,222	670	17,392
1898	3,593	15,534	2	13,587	1,650	1,206	11,355
1899	1,500	11,949	2	7,198	1,785	1,038	19,260
1900	300	5,266	2	5,756	1,891	1,168	17,609
1901	738	12,103	7	6,041	1,465	774	(d)18,423
1902	2,295	(c)	....	4,848	1,913	(c)	(d)19,611
1903	1,795	37,972	5	3,448	1,930	888	(d)18,640
1904	1,456	29,706	11	2,673	1,124	785	(d)20,890

Year.	Glass and Glassware.	Iodine. *Kg.*	Iron.				
			Ore.	Pig and Scrap.	Bars and Hoops.	Nails and Spikes.	Steel.
1895..................	1,153	2,683	1,545	8,188	19	10,408	133
1896..................	1,231	1,959	2,051	5,493	12	10,664	132
1897..................	1,432	2,395	4,242	4,631	56	9,097	167
1898..................	841	5,474	4,601	3,844	25	7,270	158
1899..................	840	16,180	12,517	6,085	337	6,089	377
1900..................	1,531	11,210	27,158	8,141	135	5,643	220
1901..................	2,142	10,000	39,173	3,250	370	6,001	179
1902..................	(c)	....	48,775	7,359	166	6,431	240
1903..................	(c)	11,417	41,575	6,350	10	6,504	200
1904..................	1,304	9,414	45,434	10,152	22	7,477	167

Year.	Nickel Ore.	Pyrites.	Silver Ore.	Stone.			
				Ashlar.	Marble and Dolomite.	Quartz.	Whetstones.
1895....................	*Nil.*	39,710	137	54,888	1,995	2,365	169
1896....................	*Nil.*	41,562	174	66,233	5,421	3,178	205
1897....................	*Nil.*	70,552	119	74,492	3,111	5,608	112
1898....................	30	67,502	79	98,692	4,267	2,244	137
1899....................	63	83,912	14	105,591	2,814	3,291	170
1900....................	272	84,604	90	101,959	3,134	3,523	136
1901....................	55	104,151	6	121,362	2,052	3,512	181
1902....................	1	105,980	*Nil.*	144,691	2,341	3,428	138
1903....................	*Nil.*	118,148	*Nil.*	165,874	4,491	4,485	170
1904....................	30	116,550	*Nil.*	191,157	3,132	6,679	169

(a) From *Tabeller vedkommende Norges Bergvaerksdrift und Tabeller vedkommende Norges Handel.* (b) Includes flowers of sulphur. (c) Returns not available. (d) Includes a small quantity of fluorspar.

PORTUGAL.

The subjoined table reports the mineral production of Portugal:

MINERAL PRODUCTION OF PORTUGAL. (*a*)
(In metric tons.)

Year.	Antimony Ore.	Arsenic.	Coal, (Anthracite) (*c*)	Coal Lignitic.	Copper. Ore.	Copper. Cement.	Copper. Pyrite.
1895	753	(*b*)	8,787	10,309	202	5,055	195,304
1896	595	(*b*)	8,743	8,000	436	3,453	207,440
1897	418	524	7,996	9,342	241	3,304	276,738
1898	245	751	10,250	12,291	290	3,149	302,686
1899	59	1,083	11,930	10,269	408	2,521	347,234
1900	38	1,031	24,066	(*b*)	(*b*)	2,948	402,870
1901	(*b*)	527	16,000	(*b*)	(*b*)	2,061	443,397
1902	68	736	11,000	5,792	655	2,205	413,714
1903	83	698	8,063	(*b*)	527	2,448	376,177
1904	31	1,370	11,844	(*b*)	(*e*)	(*b*)	463,731

Year.	Gold Ore.	Iron Ore.	Lead Ore. (Galena)	Manganese Ore.	Tin Ore and Metal.	Tungsten Ore.
1895	222.0	(*b*)	1,346	1,240	3	12
1896	(*b*)	(*b*)	1,333	1,494	6	14
1897	17.0	(*b*)	2,180	1,652	9	29
1898	6.8	2,519	3,242	907	102	59
1899	13.0	15,078	3,468	2,949	30	55
1900	(*d*)2.6	19,803	3,620	1,971	81	49
1901	(*d*)2.0	21,599	445	904	31	90
1902	(*d*)2.0	19,914	1,651	(*b*)	24	234
1903	(*d*)1.3	15,200	830	30	(*b*)	228
1904	*Nil.*	11,205	9	1,851	51	358

(*a*) From a report specially furnished THE MINERAL INDUSTRY by Señor Severiano Augusto da Fonseca Monteiro, Chief of the Department of Mines of the Ministerio das Obras Publicas except for the year 1904, which was from official Government report. (*b*) Not reported. (*c*) Consumed in the country. (*d*) *Kg.* fine metal. (*e*) Included under pyrite.

SPAIN.

The following table records the mineral and metal production of Spain:

MINERAL PRODUCTION OF SPAIN. (*a*)
(In metric tons.)

Year.	Aluminous Earths.	Antimony ore.	Arsenic Sulphide.	Asphaltum.	Asphalt Rock.	Barytes.	Cement, Hydraulic.	Coal. Anthracite.
1895	240	44	184	790	790	494	149,197	10
1896	320	54	271	1,285	1,117	345	130,738	14,895
1897	409	354	244	1,878	1,656	429	159,439	8,758
1898	505	130	111	2,354	2,383	364	164,862	20,105
1899	685	50	101	2,646	2,542	887	165,645	34,842
1900	420	30	150	2,331	4,193	833	185,811	68,427
1901	305	10	120	4,182	3,956	1,067	189,909	85,266
1902	337	67	*Nil.*	6,034	6,301	642	201,856	109,298
1903	381	42	1,088	4,675	6,277	507	245,294	108,959
1904	925	245	400	3.463	3,761	453	286,737	119,096

Year.	Coal. (*Continued.*) Bituminous.	Lignitic.	Briquettes.	Coke.	Copper Ore. Argentiferous.	Pyritic.	Copper. Fine.	Matte.	Precipitate.
1895	1,739,075	44,708	342,985	249,058	(*b*)	2,701,661	7	5,756	31,725
1896	1,852,947	55,413	343,432	288,523	(*c*) 157,365	2,200,919	6	16,378	29,873
1897	2,010,960	54,232	332,272	755,394	(*c*) 18,488	2,161,182	7	16,120	29,652
1898	2,414,127	66,422	369,418	768,151	203	2,299,444	593	16,024	29,703
1899	2,565,437	70,901	348,838	341,443	1,103	2,443,044	4	15,755	41,927
1900	2,514,545	91,133	341,156	381,000	2,006	2,714,714	5	18,159	29,652
1901	2,566,591	95,867	338,684	455,586	(*b*)	2,672,365	79	15,634	28,433
1902	2,614,010	84,242	331,957	404,503	878	2,617,776	(*b*)	.....	36,045
1903	2,587,652	104,232	339,120	433,780	3,056	2,796,733	(*b*)	.....	27,448
1904	2,903,771	100,773	307,630	432,726	(*b*)	2,624,512	(*b*)	8.117	29,494

Year.	Fluor-spar.	Gold and Silver Ore.	Iron Ore. Argentiferous.	Non-Argentiferous.	Iron and Steel. Pig.	Forged Iron.	Steel.	Kaolin. (China Clay.)
1895	27	918	572	5,514,339	179,752	48,462	56,801	836
1896	3	854	3,581	6,762,582	100,786	53,793	68,126	1,240
1897	2	2,456	5,559	7,419,768	146,940	80,894	66,007	6,294
1898	5	555	24,190	7,197,047	113,492	65,900	50,362	5,445
1899	310	(*d*) 1,110	17,139	9,397,733	113,071	40,332	112,982	2,790
1900	4	(*d*) 1,300	26,348	8,675,749	91,126	54,307	144,355	3,794
1901	(*b*)	(*d*) 1,595	27,726	7,906,517	135,600	47,085	121,023	2,220
1902	93	(*d*) 1,764	24,361	7,904,555	330,747	......	163,564	3,412
1903	4,000	(*d*) 2,681	90,996	8,304,153	380,284	......	199,642	2,578
1904	(*b*)	(*b*)	122,109	7,964,748	283.819	56,298	186,705	100,673

Year.	Lead. (Argentiferous.)		Lead. (Non-Argentiferous.)		Manganese Ore.	Mineral Paints. (Ocher.)	Phosphate Rock.	Pyrites. (Iron)
	Ore.	Metal.	Ore.	Metal.				
1895	181,433	83,978	124,195	76,808	10,162	203	1,040	60,267
1896	182,565	84,802	104,160	82,215	38,265	212	770	100,000
1897	186,692	91,258	110,469	75,112	100,566	200	2,084	100,000
1898	244,068	88,981	150,472	78,370	102,228	200	4,500	70,265
1899	184,906	70,874	123,753	91,739	104,974	100	3,510	107,386
1900	182,016	74,341	131,437	98,189	112,897	58	4,170	134,638
1901	207,188	73,895	174,376	95,399	60,325	164	4,220	133,953
1902	227,645	74,370	100,403	103,190	46,069	(b)	1,150	145,173
1903	179,858	56,687	108,660	118,422	26,194	(b)	1,124	155,739
1904	177,104	57,956	93,230	127,906	18,732	(b)	3,305	161,841

Year.	Pyrites. (arsenical.)	Quicksilver.		Salt.	Silver.		Soapstone.
		Ore.	Metal.		Ore.	Metal. *Kg.*	
1895	(b)	33,792	1,506	326,320	16,299	58,546	2,347
1896	(b)	34,959	1,524	521,751	1,230	64,554	756
1897	(b)	32,378	1,728	508,606	982	71,168	3,601
1898	230	31,361	1,691	479,358	767	76,295	2,613
1899	(b)	32,144	1,361	598,108	764	88,409	4,844
1900	515	30,216	1,095	450,041	742	140,457	8,109
1901	1,328	23,367	754	345,063	391	94,977	4,880
1902	5,648	26,037	1,425	426,434	175	96,975	542
1903	7,996	30,370	968	427,394	231	112,978	3,725
1904	3,510	27,185	1,130	543,658	303	117,418	5,165

Year.	Sulphur.		Tin Ore (*Dressed.*)	Topaz. *Kg.*	Tungsten Ore.	Zinc.		
	Crude rock.	Refined.				Ore.	Spelter.	Sheets.
1895	8,481	2,231	17	67	14	54,109	3,149	2,487
1896	26,204	1,800	(e)2,348	80	31	64,828	3,485	2,648
1897	18,805	3,500	(e)2,378	44	10	73,848	3,907	2,337
1898	34,943	3,100	4	90	37	99,836	4,300	1,731
1899	58,922	1,100	57	44	151	119,710	4,100	2,084
1900	64,364	750	47	95	1,958	86,158	2,855	2,756
1901	49,856	610	115	310	6	119,708	2,573	2,781
1902	15,442	450	12,762	*Nil.*	11	127,618	5,569	(b)
1903	38,573	1,680	330	90	*Nil.*	154,126	5,134	(b)
1904	40,389	605	229	....	60	156,329	5,887	2,913

(a) Figures for 1897 are from the Reports of the Comisión Ejecutiva de Estadistica Minera; those for 1895, 1896 and 1898 are from the official *Reports of the Junta Superior Facultativa de Minas*, Madrid. (b) Not reported. (c) Represents non-argentiferous copper ore. (d) Gold ore only. (e) Undressed tin ore.

SWEDEN.

The official statistics of mineral production, imports and exports are summarized as follows:

MINERAL PRODUCTION OF SWEDEN. (*a*)
(In metric tons.)

Year.	Alum.	Fire Clay.	Coal.	Copper.			Feldspar.	Gold-*Kg*.
				Ore.	Ingot.	Sulphate.		
1895	286	120,385	223,652	26,009	216	1,195	(*b*.)	85.2
1896	334	120,426	225,848	24,351	249	1,506	12,789	114.5
1897	131	112,283	224,343	25,207	289	1,315	19,298	113.3
1898	153	131,391	236,277	23,335	235	1,165	20,737	125.9
1899	164	129,875	239,344	22,334	179	1,287	16,017	106.2
1900	167	160,585	252,320	22,725	136	1,265	15,228	88.5
1901	121	175,876	271,509	23,660	137	1,224	13,502	62.7
1902	132	(*b*)	304,733	30,095	178	1,257	17,960	94.3
1903	140	172,718	320,390	36,687	776	1,171	19,392	50.6
1904	125	166,888	320,984	36,834	533	1,248	18,021	60.9

Year.	Iron and Steel.					Steel.		
	Ore.	Pig.	Blooms.	Bars, Rods, Sheets, etc.	Iron Sulphate	Bessemer.	Basic.	Crucible.
1895	1,904,662	462,930	188,726	284,504	94	97,320	99,259	598
1896	2,039,019	494,416	188,396	308,132	191	114,120	42,301	604
1897	2,086,119	538,197	189,633	304,537	232	107,679	165.836	691
1898	2,302,546	531,766	198,923	299,846	124	102,254	160,706	1,013
1899	2,434,606	497,727	195,331	328,999	105	91,898	179,357	1,225
1900	2,607,925	526,868	188,455	324,604	183	91,065	207,418	1,121
1901	2,793,566	528,375	164,850	269,507	140	77,231	190,877	1,088
1902	2,896,208	538,113	186,076	(*b*.)	127	84,014	201,311	1,091
1903	3,677,520	506,825	192,342	325,200	62	84,229	232,878	1,105
1904	4,083,945	528,525	189,246	324,676	148	78,577	252,832	1,162

Year.	Lead.		Manganese Ore.	Pyrites.	Silver. lead Ore.	Silver—*Kg*.	Sulphur.	Zinc Ore.
	Ore.	Pig.						
1895	7	1,256	3,117	221	12,045	1,188	(*b*)	31,349
1896	14	1,530	2,056	1,009	15,381	2,082	77	44,041
1897	99	1,480	2,749	517	10,068	2,218	(*b*)	56,636
1898	50	1,559	2,358	386	6,743	2,033	50	61,627
1899	35	1,606	2,622	150	5,730	2,290	(*b*)	65,159
1900	85	1,424	2,651	179	5,300	1,927	70	61,044
1901	56	988	2,271	*Nil*.	11,366	1,557	(*b*)	48,630
1902	63	843	2,850	*Nil*.	9,378	1,365	74	48,783
1903	25	678	2,244	7,793	9,792	1,005	(*b*)	62,927
1904	55	589	2,297	15,957	8,187	651	35	57,634

(*a*) From *Bidrag till Sveriges Officiela Statistik Bergshandteringen*. (*b*) Not reported.

MINERAL IMPORTS OF SWEDEN. (*a*)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Alum.	Aluminum Sulphate.	Ammonium.					Antimony (crude).	Arsenious Acid.
			Carbonate.	Chloride.	Hydrate.	Nitrate.	Sulphate.		
1895	93	348	74	84	76	11	39	81	36
1896	75	629	79	88	81	11	88	63	33
1897	103	733	109	110	59	42	67	58	33
1898	136	968	99	101	105	12	81	53	33
1899	158	866	89	112	110	12	181	59	12
1900	133	1,197	141	99	100	5	227	85	22
1901	346	1,192	131	113	92	1	285	50	12
1902	250	1,430	127	145	130	13	241	59	14
1903	302	1,082	113	133	150	47	197	54	21
1904	87	1,245	154	208	115	41	254	67	17

Year.	Asbestos. (c)	Asphalt.	Barytes	Borax.	Boric Acid.	Bromine and Bromides. *Kg.*	Cement	Chalk. White. Unground. *Hectoliters.*
1895	68	3,256	240	111	54	3,999	4,270	4,466
1896	116	4,092	298	128	73	4,334	2,901	6,148
1897	119	5,458	270	175	56	5,549	1,826	14,368
1898	112	5,409	299	196	75	5,401	1,656	7,016
1899	567	6,286	292	190	65	4,914	1,363	16,079
1900	763	5,676	411	194	66	6,084	1,941	12,059
1901	178	4,524	295	253	68	6,602	2,868	13,569
1902	213	5,779	....	242	71	7,278	9,822	11,583
1903	217	5,957	....	240	71	7,419	11,145	41,868
1904	356	6,243	....	299	77	10,128	10,526	10,115

Year.	Coal.	Copper, also Alloys.	Emery.	Gold. Bars and Mfres. *Kg.*	Gold. Specie.	Graphite.	Gypsum.	Iron (crude.)
1895	1,911,629	2,930	125	89	$11,600	134	4,112	40,582
1896	1,991,760	4,037	104	1,161	608	135	4,940	34,549
1897	2,240,247	4,944	128	4,267	948	158	7,260	89,606
1898	2,392,451	5,227	131	3,998	2,396	167	7,979	76,832
1899	3,047,618	4,740	125	362	9,774	162	6,457	68,909
1900	3,033,885	4,745	136	3,365	98,905	213	6,794	82,957
1901	2,793,309	5,153	169	1,454	736,852	180	6,589	66,131
1902	2,911,286	6,890	147	945	Nil.	(b)	6,754	43,828
1903	3,192,990	6,109	132	89	965,346	(b)	8,795	49,411
1904	3,367,826	7,367	221	1,400	1,207,187	(b)	8,868	90,102

Year.	Lead.	Lime *Hectoliters.*	Litharge.	Phosphorus. *Kg.*	Platinum. *Kg.*	Porcelain	Potassium. Chloride.	Potassium. Cyanide. *Kg.*	Potassium. Hydrate.	Potassium. Carbonate.
1895	1,624	4,436	117	71,407	42	277	561	1,457	91	1,979
1896	1,911	7,768	150	52,482	34	327	241	2,122	285	1,933
1897	2,098	20,050	199	57,972	63	362	363	2,922	1,381	1,432
1898	2,139	23,079	160	66,466	49	298	259	2,604	1,451	1,112
1899	2,125	34,343	177	59,989	59	346	225	2,313	1,266	1,231
1900	2,067	25,047	148	67,557	99	382	364	2,221	1,915	1,257
1901	1,991	12,204	165	70,672	172	386	260	2,658	1,435	1,266
1902	2,509	6,579	172	68,441	130	....	222	2,950	1,720	1,238
1903	2,644	6,449	237	112,659	116	....	245	3,294	2,034	1,150
1904	2,849	13,388	213	47,421	84	396	214	3,237	2,234	1,184

Year.	Quicksilver. *Kg.*	Salt. Crude.	Salt. Refined.	Silver. Scrap and Mfres.	Silver. Specie.	Sodium. Carbonate.	Sodium. Hydrate.	Sodium. Nitrate. (d)	Sodium. Sulphate. (e)
1895	6,318	106,820	2,535	3,050	$114,340	12,711	1,043	9,388	7,420
1896	5,194	84,629	3,673	7,375	204,691	11,425	908	12,518	8,486
1897	3,125	87,050	3,055	20,557	136,823	14,625	625	12,531	11,384
1898	2,631	85,246	2,188	21,696	191,766	11,917	575	15,419	11,544
1899	4,210	98,417	3,166	11,565	156,707	13,323	929	15,006	15,140
1900	3,629	70,302	3,098	11,559	62,315	12,680	1,038	14,245	15,590
1901	5,958	79,038	3,072	7,476	78,416	13,669	800	17,614	15,494
1902	4,866	82,439	3,037	4,853	74,826	....	1,623	15,553	18,924
1903	5,043	88,139	3,419	11,259	90,366	....	1,426	20,616	16,120
1904	5,768	84,237	4,615	19,034	86,891	11,898	2,112	19,776	17,596

SWEDEN.

The official statistics of mineral production, imports and exports are summarized as follows:

MINERAL PRODUCTION OF SWEDEN. (*a*)
(In metric tons.)

Year.	Alum.	Fire Clay.	Coal.	Copper.			Feldspar.	Gold-*Kg*.
				Ore.	Ingot.	Sulphate.		
1895	286	120,385	223,652	26,009	216	1,195	(*b*.)	85.2
1896	334	120,426	225,848	24,351	249	1,506	12,789	114.5
1897	131	112,283	224,343	25,207	289	1,315	19,298	113.3
1898	153	131,391	236,277	23,335	235	1,165	20,737	125.9
1899	164	129,875	239,344	22,334	179	1,287	16,017	106.2
1900	167	160,585	252,320	22,725	136	1,265	15,228	88.5
1901	121	175,876	271,509	23,660	137	1,224	13,502	62.7
1902	132	(*b*)	304,733	30,095	178	1,257	17,960	94.3
1903	140	172,718	320,390	36,687	776	1,171	19,392	50.6
1904	125	166,888	320,984	36,834	533	1,248	18,021	60.9

Year.	Iron and Steel.					Steel.		
	Ore.	Pig.	Blooms.	Bars, Rods, Sheets, etc.	Iron Sulphate	Bessemer.	Basic.	Crucible.
1895	1,904,662	462,930	188,726	284,504	94	97,320	99,259	598
1896	2,039,019	494,416	188,396	308,132	191	114,120	42,301	604
1897	2,086,119	538,197	189,633	304,537	232	107,679	165.836	691
1898	2,302,546	531,766	198,923	299,846	124	102,254	160,706	1,013
1899	2,434,606	497,727	195,331	328,999	105	91,898	179,357	1,225
1900	2,607,925	526,868	188,455	324,604	183	91,065	207,418	1,121
1901	2,793,566	528,375	164,850	269,507	140	77,231	190,877	1,088
1902	2,896,208	538,113	186,076	(*b*.)	127	84,014	201,311	1,091
1903	3,677,520	506,825	192,342	325,200	62	84,229	232,878	1,105
1904	4,083,945	528,525	189,246	324,676	148	78,577	252,832	1,162

Year.	Lead.		Manganese Ore.	Pyrites.	Silver. lead Ore.	Silver—*Kg*.	Sulphur.	Zinc Ore.
	Ore.	Pig.						
1895	7	1,256	3,117	221	12,045	1,188	(*b*)	31,349
1896	14	1,530	2,056	1,009	15,381	2,082	77	44,041
1897	99	1,480	2,749	517	10,068	2,218	(*b*)	56,636
1898	50	1,559	2,358	386	6,743	2,033	50	61,627
1899	35	1,606	2,622	150	5,730	2,290	(*b*)	65,159
1900	85	1,424	2,651	179	5,300	1,927	70	61,044
1901	56	988	2,271	*Nil*.	11,366	1,557	(*b*)	48,630
1902	63	843	2,850	*Nil*.	9,378	1,365	74	48,783
1903	25	678	2,244	7,793	9,792	1,005	(*b*)	62,927
1904	55	589	2,297	15,957	8,187	651	35	57,634

(*a*) From *Bidrag till Sveriges Officiela Statistik Bergshandteringen*. (*b*) Not reported.

MINERAL IMPORTS OF SWEDEN. (*a*)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Alum.	Aluminum Sulphate.	Ammonium.					Antimony (crude).	Arsenious Acid.
			Carbonate.	Chloride.	Hydrate.	Nitrate.	Sulphate.		
1895	93	348	74	84	76	11	39	81	36
1896	75	629	79	88	81	11	88	63	33
1897	103	733	109	110	59	42	67	58	33
1898	136	968	99	101	105	12	81	53	33
1899	158	866	89	112	110	12	181	59	12
1900	133	1,197	141	99	100	5	227	85	22
1901	346	1,192	131	113	92	1	285	50	12
1902	250	1,430	127	145	130	13	241	59	14
1903	302	1,082	113	133	150	47	197	54	21
1904	87	1,245	154	208	115	41	254	67	17

Year.	Asbestos. (*c*)	Asphalt.	Barytes	Borax.	Boric Acid.	Bromine and Bromides. *Kg.*	Cement	Chalk. White. Unground. *Hectoliters.*
1895	68	3,256	240	111	54	3,999	4,270	4,466
1896	116	4,092	298	128	73	4,334	2,901	6,148
1897	119	5,458	270	175	56	5,549	1,826	14,368
1898	112	5,409	299	196	75	5,401	1,656	7,016
1899	567	6,286	292	190	65	4,914	1,363	16,079
1900	763	5,676	411	194	66	6,084	1,941	12,059
1901	178	4,524	295	253	68	6,602	2,868	13,569
1902	213	5,779	....	242	71	7,278	9,822	11,583
1903	217	5,957	....	240	71	7,419	11,145	41,868
1904	356	6,243	....	299	77	10,128	10,526	10,115

Year.	Coal.	Copper, also Alloys.	Emery.	Gold. Bars and Mfres. *Kg.*	Gold. Specie.	Graphite.	Gypsum.	Iron (crude.)
1895	1,911,629	2,930	125	89	$11,600	134	4,112	40,582
1896	1,991,760	4,037	104	1,161	608	135	4,940	34,549
1897	2,240,247	4,944	128	4,267	948	158	7,260	89,606
1898	2,392,451	5,227	131	3,998	2,396	167	7,979	76,832
1899	3,047,618	4,740	125	362	9,774	162	6,457	68,909
1900	3,033,885	4,745	136	3,365	98,905	213	6,794	82,957
1901	2,793,309	5,153	169	1,454	736,852	180	6,589	66,131
1902	2,911,286	6,890	147	945	Nil.	(*b*)	6,754	43,828
1903	3,192,990	6,109	132	89	965,346	(*b*)	8,795	49,411
1904	3,367,826	7,367	221	1,400	1,207,187	(*b*)	8,868	90,102

Year.	Lead.	Lime *Hectoliters.*	Litharge.	Phosphorus. *Kg.*	Platinum. *Kg.*	Porcelain	Potassium. Chloride.	Potassium. Cyanide. *Kg.*	Potassium. Hydrate.	Potassium. Carbonate.
1895	1,624	4,436	117	71,407	42	277	561	1,457	91	1,979
1896	1,911	7,768	150	52,482	34	327	241	2,122	285	1,933
1897	2,098	20,050	199	57,972	63	362	363	2,922	1,381	1,432
1898	2,139	23,079	160	66,466	49	298	259	2,604	1,451	1,112
1899	2,125	34,343	177	59,989	59	346	225	2,313	1,266	1,231
1900	2,067	25,047	148	67,557	99	382	364	2,221	1,915	1,257
1901	1,991	12,204	165	70,672	172	386	260	2,658	1,435	1,266
1902	2,509	6,579	172	68,441	130	....	222	2,950	1,720	1,238
1903	2,644	6,449	237	112,659	116	....	245	3,294	2,034	1,150
1904	2,849	13,388	213	47,421	84	396	214	3,237	2,234	1,184

Year.	Quicksilver. *Kg.*	Salt. Crude.	Salt. Refined.	Silver. Scrap and Mfres.	Silver. Specie.	Sodium. Carbonate.	Sodium. Hydrate.	Sodium. Nitrate. (*d*)	Sodium. Sulphate. (*e*)
1895	6,318	106,820	2,535	3,050	$114,340	12,711	1,043	9,388	7,420
1896	5,194	84,629	3,673	7,375	204,691	11,425	908	12,518	8,486
1897	3,125	87,050	3,055	20,557	136,823	14,625	625	12,531	11,384
1898	2,631	85,246	2,188	21,696	191,766	11,917	575	15,419	11,544
1899	4,210	98,417	3,166	11,565	156,707	13,323	929	15,006	15,140
1900	3,629	70,302	3,098	11,559	62,315	12,680	1,038	14,245	15,590
1901	5,958	79,038	3,072	7,476	78,416	13,669	800	17,614	15,494
1902	4,866	82,439	3,037	4,853	74,826	....	1,623	15,553	18,924
1903	5,043	88,139	3,419	11,259	90,366	....	1,426	20,616	16,120
1904	5,768	84,237	4,615	19,034	86,891	11,898	2,112	19,776	17,596

Year.	Pyrites.	Salt.	Silica. (chert and flint.)	Silver. *Kg.*	Stone.			
					Granite.	Limestone.(*d*)	Sandstone.	Slate.
1896	10,177	2,054,715	109,694	8,828	1,784,925	11,187,532	4,579,869	596,324
1897	10,752	1,933,949	95,209	7,750	1,876,880	11,179,580	5,043,535	618,941
1898	12,302	1,908,723	83,370	6,575	1,905,830	12,172,267	5,325,988	679,461
1899	12,426	1,945,531	69,955	5,969	4,785,284	12,499,736	5,296,026	650,077
1900	12,484	1,873,601	78,971	5,964	4,709,997	12,099,940	5,101,868	595,428
1901	10,405	1,812,180	132,700	5,452	5,131,787	11,363,202	5,199,234	496,756
1902	9,315	1,893,881	100,938	4,560	5,554,696	12,368,196	5,571,121	525,665
1903	9,794	1,917,274	74,355	5,440	5,512,605	12,419,120	5,496,312	540,143
1904	10,452	1,921,899	66.300	4,967	6,084,642	12,235,825	5,391,265	572,181
1905	12,381	1,920,149	71,808	(*b*)	6,052,210	12,701,808	5,729,799	523,892

Year.	Strontium Sulphate	Tin.		Tungsten Ore.	Uranium Ore.	Zinc.	
		Ore, Dressed.	Block.			Ore.	Spelter.
1896	18,331	7,786	4,915	44	36	19,588	7,224
1897	15,227	7,234	4,524	127	30	18,586	7,162
1898	13,148	7,498	4,722	331	26	23,929	8,711
1899	12,831	6,494	4,077	96	7	23,505	8,837
1900	9,270	6,911	4,337	9	42	25,070	9,214
1901	16,923	7,407	4,634	21	80	23,967	8,555
1902	32,799	7,681	4,462	9	53	25,462	9,275
1903	23,209	7,500	4,351	276	6	25,287	9,430
1904	18,460	6,849	4,198	164	Nil.	28,097	10,427
1905	14,523	7,316	(*b*)	174	105	24,025	(*b*)

(*a*) From *Mineral Statistics of the United Kingdom.* (*b*) Not reported. (*c*) Bog ore, which is raised in Ireland, is an ore of iron, used principally for purifying gas. (*d*) Does not include chalk. (*e*) Includes China clay, potters' clay, and fuller's earth.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Alkali.	Asphaltum.	Borax.	Brass and Bronze Mfrs.	Clay Products, Porcelain and Earthenware.	Coal, Coke and Pat. Fuel.	Copper.		
							Ore.	Regulus and Precipitate	Wrought, Unwrought and Old.
1896	8,949	29,069	(*b*)	2,075	16,444	16,000	88,732	92,252	66,405
1897	11,557	44,541	(*b*)	2,129	17,847	9,605	83,916	90,008	62,055
1898	12,179	46,398	1,255	2,357	16,405	11,191	91,141	76,201	70,018
1899	12,078	59,073	3,076	1,988	18,341	1,777	130,611	84,015	60,502
1900	16,360	53,061	15,667	2,335	18,329	10,112	102,365	89,123	72,223
1901	(*c*)13,429	74,694	15,710	2,469	20,754	7,685	102,503	93,338	68,809
1902	(*c*)26,292	65,896	13,390	3,403	19,029	3,331	90,007	74,684	92,349
1903	(*c*)14,321	(*b*)	11,959	(*b*)	18,576	3,535	85,644	77,884	64,591
1904	(*c*) 14,325	(*b*)	16,012	(*b*)	18,980	2,812	80,771	67,739	90,717
1905	(*c*)16.593	(*b*)	11,552	(*b*)	20,076	49,277	94,198	70.235	71,294

Year.	Diamonds.	Glass. All Kinds.	Gold.			Iron and Steel.		
			Ore.	Bullion and Specie.	Leaves. *Number.*	Iron Ore.	Pig Iron.	Scrap.
1896	$22,995,830	$12,537,885	$604,015	$122,341,685	70,483,026	5,525,320	(*f*)108,152	15,578
1897	22,176,300	15,034,555	736,415	154,044,290	68,173,400	6,064,179	(*f*)160,531	20,735
1898	22,619,075	16,423,915	1,393,175	218,614,800	63,632,700	5,555,889	(*f*)162,075	24,619
1899	20,597,910	16,044,400	761,755	162,667,485	49,108,570	7,168,061	(*f*)174,159	32,427
1900	17,168,180	15,997,555	1,432,970	130,954,365	54,346,025	6,398,639	178,199	31,687
1901	24,385,210	17,647,635	2,747,195	103,578,140	59,048,856	5,637,670	198,536	44,721
1902	26,901,950	18,484,960	2,688,170	108,145,245	59,334,023	6,542,793	226,708	39,584
1903	(*b*)	18,636,475	3,590,095	143,286,965	(*b*)	6,417,188	132,364	17,051
1904	(*b*)	16,901,500	4,259,360	169,382,940	(*b*)	6,198,368	132,494	19,326
1905	(*b*)	16,992,835	(*b*)	192,839,475	(*b*)	6,888,061	128,183	23,569

Year.	Iron and Steel. (*Continued.*) Puddled and Wrought.	Sheets and Plates.	Rails.	Strips and Wire Rods.	Nails, Screws. Rivets, Bolts.	Steel Ingots, Blooms, Billets, etc.	Steel Bars, Shapes, Beams and Pillars	Mnfrs. Unenumerated (*k*)
1896	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	17,771	(*h*)	$22,055,335
1897	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	40,628	(*h*)	27,894,295
1898	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	40,875	(*h*)	33,379,160
1999	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	78,257	(*h*)	39,527,075
1900	189,891	(*g*)	38,636	(*g*)	(*g*)	182,210	94,667	17,861,660
1901	102,811	(*g*)	55,809	(*g*)	(*g*)	185,810	124,648	18,252,435
1902	178,425	(*g*)	48,942	(*g*)	45,095	285,494	129,743	14,424,835
1903	196,084	73,079	74,939	35,574	51,888	278,441	343,259	9,007,990
1904	109,289	69,552	40,438	38,214	50,649	531,069	219,510	8,630,380
1905	110,576	69,831	34,439	60,318	55,331	613,612	148,995	7,779,395

Year.	Lead. Ore.	Lead. Pig and Sheet.	Manganese Ore.	Mica, Sheet.	Mica and Talc.	Paraffin.	Petroleum. *Liters.*	Phosphate Rock.
1896	57,172	170,484	162,542	431	1,753	36,850	862,960,772	295,975
1897	32,818	170,121	158,825	412	1,683	39,284	842,920,307	330,335
1898	44,457	197,591	156,390	517	1,398	48,104	829,995,751	334,884
1899	30,263	201,551	261,740	519	6,025	54,712	908,107,248	426,830
1900	21,566	198,416	270,098	469	7,952	50,033	965,167,850	361,309
1901	29,944	221,549	195,736	(*b*)	7,117	42,643	960,650,967	360,568
1902	25,838	235,522	237,066	1,078	6,127	52,023	1,078,095,152	370,697
1903	18,923	232,939	235,574	(*b*)	(*b*)	49,163	1,299,570,625	398,997
1904	8,748	250,452	208,458	(*b*)	(*b*)	42,882	1,373,488,176	425,978
1905	......	233,214	289,827	(*b*)	(*b*)	41,247	1,364,301,583	427,762

Year.	Paints, Mineral.	Platinum Wrought and Unwrought. *Kg.*	Potassium Nitrate.	Pyrites of Iron and Copper.	Quicksilver.	Silver Ore. (*e*)	Sodium Nitrate.
1896	$4,950,635	1,748	17,310	598,480	1,604	$6,483,680	108,148
1897	5,049,225	2,257	16,744	633,009	1,862	7,149,210	107,525
1898	5,619,835	3,389	13,323	665,544	1,856	5,729,525	132,412
1899	5,842,063	5,404	12,635	712,393	1,759	5,162,750	163,387
1900	6,747,800	5,027	12,798	752,605	1,113	5,154,430	143,461
1901	6,489,710	4,917	12,115	664,041	1,202	5,309,920	108,822
1902	6,367,320	3,027	11,526	620,948	1,129	5,383,515	116,791
1903	7,025,050	(*b*)	9,425	747,714	1,187	6,596,045	118,582
1904	6,718,095	(*b*)	12,277	754,722	1,130	8,271,480	122,454
1905	....	(*b*)	8,260	709,926	1,158	10,426,570	106,107

Year.	Stone and Marble, Hewn or Mfrd.	Sulphur.	Tin. Ore.	Tin. Block, Ingot, Bars or Slabs.	Zinc. Ore.	Zinc. Spelter.	Zinc. Mnfres.
1896	683,977	22,990	4,950	38,989	21,421	77,861	21,339
1897	752,345	22,811	5,345	27,214	25,238	70,929	21,395
1898	883,699	19,642	5,710	20,665	53,945	78,761	21,613
1899	905,432	21,906	6,324	27,608	38,143	71,068	21,521
1900	961,492	22,993	7,449	33,648	42,755	61,504	21,751
1901	1,159,276	22,440	10,690	35,397	38,660	68,633	21,343
1902	1,192,023	23,863	12,255	35,713	45,312	89,688	21,717
1903	1,219,782	21,313	12,473	36,076	41,009	86,539	23,118
1904	1,329,502	17,629	15,734	39,932	54,438	90,088	22,788
1905	1,217,652	18,163	....	40,391	....	92,261	20,013

(*b*) Not reported. (*c*) Classified as soda compounds since 1901 (*d*) Includes machinery and mill work. (*e*) Includes the value of silver in argentiferous ore and metal. (*f*) Includes puddled iron. (*g*) Not separately enumerated. (*h*) Former returns not available. (*k*) Prior to 1900 many manufactures were not reported separately.

MINERAL EXPORTS OF THE UNITED KINGDOM—DOMESTIC PRODUCTS. (a)
(In metric tons or dollars; £1=$5.)

Year.	Ammonium Sulphate.	Bleaching Materials.	Brass and Mnfrs.	Cement.	Clay. Unmanufactured.	Clay products. (c)	Coal.
1896	130,130	(b)	6,070	359,429	(b)	$10,847,690	33,474,843
1897	155,449	(b)	5,713	398,023	(b)	8,629,425	35,919,965
1898	139,806	(b)	5,418	331,648	(b)	8,253,640	35,619,365
1899	142,577	(b)	5,797	359,273	(b)	9,314,255	41,839,217
1900	147,528	57,478	6,131	365,742	(b)	10,193,970	46,845,739
1901	154,282	46,912	5,905	318,216	(b)	9,963,985	42,547,114
1902	165,213	40,939	6,378	308,104	463,309	9,497,995	43,849,591
1903	....	49,415	7,646	406,388	490,157	10,879,940	45,669,258
1904	....	35,289	8,860	390,736	527,013	10,531,620	46,995,636
1905	....	42,526	12,526	463,863	573,706	10,487,585	48,236,334

Year.	Coke.	Patent Fuel.	Supplied to Steamers.	Coal Products (d)	Copper.			
					Ingot.	Mixed or Yellow Metal	Mnfrs.	Sulphate.
1896	687,640	(b)	10,096,302	$9,168,630	23,698	11,428	15,593	53,464
1897	993,980	(b)	10,623,050	8,340,420	21,252	11,192	15,275	60,326
1898	782,053	(b)	11,444,431	7,624,740	27,102	10,452	13,765	52,573
1899	881,172	(b)	12,422,429	7,712,965	32,449	7,038	11,231	40,822
1900	1,001,131	(b)	11,940,353	9,058,220	18,300	8,940	10,765	43,601
1901	820,594	1,098,459	13,804,222	5,756,265	26,935	9,252	11,156	36,601
1902	669,664	1,067,060	15,390,485	5,991,025	21,658	13,314	14,075	43,995
1903	728,957	970,449	17,068,646	7,290,825	23,723	14,425	16,975	54,307
1904	779,060	1,257,589	17,465,954	6,879,400	14,791	16,704	18,467	71,367
1905	786,496	1,126,190	17,674,484	6,738,370	17,004	8,706	18,038	55,219

Year.	Fertilizers.	Glass. All kinds.	Ore.	Iron.				
				Pig.	Scrap.	Cast Iron and Mnfrs.	Wrought Iron, Shapes and Mnfrs.	Rails.
1896	(e)	4,462,445	(e)	(f)1,077,128	129,463	(b)	180,973	(g)759,625
1897	(e)	4,359,535	(e)	(f)1,219,958	99,259	(b)	170,285	(g)795,983
1898	(e)	4,410,205	(e)	(f)1,058,973	86,602	(b)	152,911	(g)619,976
1899	(e)	4,578,415	(e)	(f)1,401,365	118,262	(b)	161,679	(g)601,266
1900	$12,041,450	5,172,910	(e)	(f)1,450,365	96,567	(b)	159,677	379,939
1901	11,987,555	5,285,775	(e)	(f) 852,609	86,559	(b)	119,962	474,073
1902	13,862,875	5,489,650	4,062	1,120,207	104,890	(b)	(b)	(b)
1903	14,254,340	5,512,470	4,534	1,082,426	143,929	62,249	217,139	613,741
1904	14,869,250	5,049,495	6,706	823,909	166,010	49,004	173,233	533,895
1905	14,425,085	5,558,820	14,664	997,601	151,619	41,729	186,340	555,390

Year.	Iron. (*Continued.*)								Lead. Pig and Mnfrs.
	Wire and Mnfrs. of.	Plates and Sheets.	Galvanized Sheets.	Black Plates for Tinning.	Tinned Plates.	Steel Ingots, Billets, Blooms, etc	Steel Shapes Beams and Pillars.	Total Iron and Steel and Mnfrs. of.	
1896	57,008	122,660	248,348	49,179	271,234	302,198	(b)	3,607,204	41,874
1897	52,471	120,868	231,319	59,663	276,260	304,249	(b)	3,750,122	40,911
1898	44,954	102,638	230,219	59,289	255,797	290,182	(b)	3,299,326	38,684
1899	50,041	111,773	242,167	86,936	260,735	333,837	(b)	3,777,098	40,923
1900	39,104	39,157	251,203	66,810	278,338	313,383	(b)	3,602,083	36,576
1901	48,107	36,418	254,290	52,217	275,661	217,236	(b)	2,944,083	38,166
1902	(b)	(b)	336,572	58,245	317,201	306,152	(b)	3,529,223	33,537
1903	60,800	165'672	357,665	66,279	297,485	13,427	159,330	3,621,635	36,152
1904	61,894	154,774	391,608	63,467	365,262	4,324	176,232	3,315,047	35,600
1905	82,519	207,866	413,533	69'937	360,630	8,735	219,491	3,781,059	42,265

Year.	Salt.	Sodium.				Tin. Block.	Zinc.		
		Soda Ash.	Carbonate and Bicarbonate.	Hydrate.	Sulphate.		Ore.	Spelter.	Mnfrs.
1896	671,510	(h)245,938	(i)	(i)	(i)	6,299	3,368	8,605	1,034
1897	680,477	(h)252,736	(i)	(i)	(i)	5,050	6,072	6,951	1,047
1898	698,882	(h)191,578	(i)	(i)	(i)	5,557	6,483	7,577	1,227
1899	638,213	(h)193,492	(i)	(i)	(i)	4,785	8,171	5,492	1,249
1900	556,704	(h)185,783	(i)	(i)	(i)	5,713	13,913	7,136	1,159
1901	627,078	58,412	22,161	50,624	26,057	5,584	13,981	7,512	1,256
1902	624,752	59,894	24,654	61,658	35,672	6,210	16,717	6,756	1,345
1903	594,300	58,605	23,574	59,725	45,630	6,349	15,659	8,102	(k)
1904	632,605	61,327	25,252	61,985	40,324	5,953	14,606	7,993	(k)
1905	588,389	67,678	28,425	68,675	33,681	7,741	....	7,451	(k)

(a) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (b) Not reported. (c) Comprises porcelain and earthenware. (d) Including naphtha, paraffin, paraffin oil and petroleum. (e) Previous reports not available. (f) Includes puddled iron. (g) Includes railroad material of all kinds. (h) Includes all soda compounds; not separately enumerated previous to 1901. (i) Included under soda ash. (k) Included under spelter.

UNITED STATES.

Of the following three tables, the first records the imports of foreign mineral and metal products into the United States, whether dutiable or duty free; the second shows the exports of materials produced in the United States; and the third reports the re-exports of products of foreign origin. The statement of production in the United States is given on an early page in this volume. The following statistics are those reported by the Bureau of Statistics of the Department of Commerce and Finance.

IMPORTS (*a*)

Year.	Aluminum. Crude. Lb.	Kg.	Value.	Value per Lb.	Mfrd. Value.	Ammonium Sulphate. Lb.	Metric Tons.	Value.	Value per Lb.
1900	256,559	116,374	$44,455	$0.172	$3,111	24,024,188	10,897	$591,937	$0.025
1901	564,803	251,657	104,168	.186	5,580	31,711,085	14,384	728,085	.023
1902	745,217	338,028	215,032	.290	3,645	35,535,558	16,119	858,036	.024
1903	498,655	226,190	139,298	.279	4,273	29,104,817	13,199	765,230	.026
1904	515,416	234,293	128,350	.249	478	39,859,690	18,077	1,058,981	.027
1905	530,429	240,284	106,108	.200	33	15,512,399	7,038	416,048	.027

Year.	Antimony. Lb.	Metric Tons.	Value.	Value per Lb.	Antimony Ore. Lb.	Metric Tons.	Value.	Value per Lb.
1900	3,632,843	1,648	$285,749	$0.079	6,035,734	2,738	$78,581	$0.013
1901	3,674,923	1,667	255,346	.069	1,731,756	786	24,256	.014
1902	5,742,703	2,605	347,899	.061	1,639,043	743	29,476	.018
1903	5,125,515	2,325	279,957	.054	2,673,142	1,213	51,489	.019
1904	4,056,299	1,840	235,401	.058	2,487,602	1,129	50,362	.020
1905	5,737,891	2,603	431,774	.075	1,976,694	897	52,868	.027

Year.	Asbestos. Crude Value.	Mfd. Value.	Total Value.	Asphaltum. Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	$331,796	$24,155	$355,951	113,557	115,374	$404,921	$3.57
1901	667 087	24,741	691,828	132,079	134,192	516,515	3.85
1902	729,421	33,013	762,434	139,944	142,183	439,570	3.09
1903	657,269	32,058	689,327	167,554	170,235	514,051	3.06
1904	700,572	51,290	751,862	119,575	121,489	510,524	4 27
1905	776,362	70,117	846,479	86,798	88,187	382,667	4 41

Year.	Arsenic. (b)				Barytes.				Bauxite.			
	Lb.	Metric Tons.	Value.	Value per	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900....	(f)	.....	.....	.....	(f)	.....	.....	.....	8,656	8,795	$32,967	$3.81
1901....	(f)	.....	.....	.....	(f)	.....	.....	.....	17,866	18,153	66,107	3.70
1902....	(f)	.....	.....	.....	(f)	.....	.....	.....	15,790	16,043	54,410	3.45
1903....	7,391,566	3,241	$256,097	$0.036	6,344	6,446	$22,777	$3.59	14,889	15,127	49,684	3.34
1904....	6,391,566	2,900	226,481	0.036	6,689	6,796	27,463	4.11	15,475	15,723	49,577	3.20
1905....	6,444,083	2,924	219,198	0.034	7,879	8,005	36,796	4.67	11,726	11,914	46,517	3.96

Year.	Brass and Mfrs. of, Value.	Chloride of Lime.				Cement.			
		Lb.	Metric Tons.	Value.	Value per Lb.	Barrels. (c)	Metric Tons.	Value.	Value per Bbl.
1900......	$20,113	132,520,478	60,111	$1,524,205	$0.012	2,386,684	433,037	$3,330,453	$1.40
1901......	35,976	120,611,346	54,709	1,673,190	.014	944,892	170,431	1,305,692	1.38
1902......	51,626	112,374,478	50,973	1,456,435	.013	1,994,790	361,932	2,581,883	1.29
1903......	206,905	113,285,240	51,586	912,843	.008	2,317,951	420,569	3,027,111	1.30
1904......	366,220	87,909,168	39,876	707,174	.008	1,046,404	189,910	1,382,913	1.32
1905......	1,331,827	104,919,462	47,604	843,285	.008	846,577	153,644	1,102,041	1.30

Year.	Chrome Ore.				Bismuth.				Clays and Earths.			
	Long Tons.	Metric Tons.	Value.	Value per L.T	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900..........	17,572	17,823	$305,001	$17.38	....	......	....	......	146,524	148,868	$966,579	$6.59
1901..........	20,112	20,434	363,108	18.04	....	......	....	......	181,013	183 589	1,176,633	6.51
1902..........	39,570	40,203	582,597	14.72	....	......	....	......	191,764	194,832	1,228,945	6.41
1903..........	22,932	23,299	302,025	13.17	147,324	66,826	235,199	$1.59	203,173	206,424	1,264,544	6.22
1904..........	24,227	24,615	348,527	14.38	147,712	67,002	268,837	1.82	191,853	194,923	1,168,552	6.09
1905..........	54,434	55,305	725,301	13.32	148,589	67,459	318,007	2.14	321,641	326,787	1,363,760	4.24

Year.	Coal, Anthracite.			Lead.		Coal, Bituminous.			Total Coal.	
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Value.
1900.....	118	120	$549	$4.65	1,909,258	1,939,806	$5,019,553	$2.63	1,909,366	5,020,102
1901.....	286	291	1,844	6.45	1,919,962	1,950,681	5,291,429	2.75	1,920,248	5,293,273
1902.....	73,006	74,174	323,517	4.43	2,478,375	2,518,029	7,012,674	2.84	2,551,381	7,339,791
1903.....	151,023	153,439	675,623	4.47	3,295,379	3,348,105	9,329,221	2.83	3,446,402	10,004,844
1904.....	72,526	73,686	220,665	3.04	1,556,149	1,581,047	3,915,613	2.52	1,628,675	4,136,278
1905.....	34,262	34,810	107,394	3.13	1,618,581	1,644,478	3,908,877	2.42	1,652,843	4,016,271

Year.	Coke.				Cobalt Oxide.				Copper, Ore and Matte.			
	Long Tons.	Metric Tons.	Value.	Value pr L.T.	Lb.	Kg.	Value.	Value pr Lb.	Long Tons.	Metric Tons.	Value	Value per L.T.
1900..........	103,175	104,826	$371,341	$3.60	54,073	24,527	$88,651	$1.64	54,329	55,201	$5,195,010	$92.23
1901..........	172,729	173,893	266,078	3.67	71,969	32,645	134,208	1.86	96,047	97,584	14,692,645	152.99
1902..........	107,437	109,156	423,774	4.05	79,984	36,281	151,115	1.89	181,566	184,470	8,695,780	47.89
1903..........	127,479	129,519	437,625	3.43	73,350	33,272	145,264	1.98	284,912	289,471	3,177,582	11.15
1904..........	161,476	164,060	648,520	4.01	42,352	19,211	86,925	2.05	268,234	272,527	4,308,410	16.06
1905..........	181,376	184,278	796,544	4.39	70,048	31,802	139,377	1.99	296,251	300,991	5,765,238	19.46

Year.	Copper, Ingots, Old, etc.				Copper Mfrs.	Total Copper.	Cryolite.			
	Lb.	Metric Tons.	Value.	Value per Lb.			Long Tons.	Metric Tons.	Value.	Value per L. T.
1900..........	68,796,808	31,206	$10,557 870	$0.153	$23,390	$15,776,270	5,437	5,524	$72,763	$13.38
1901..........	73,826,406	33,488	11,812,216	.160	24,775	26,529,636	5,383	5,469	70,886	13.17
1902..........	103,129,568	46,778	13,051,159	.126	52,464	21,799,403	6,188	6,287	85,640	13.83
1903..........	136,707,995	62,011	17,262,148	.126	31,624	20,471,354	7,708	7,831	102,879	13.35
1904..........	142,344,433	64,567	18,374,959	.129	37,913	22,721,282	959	974	13,708	14.30
1905..........	160,619,385	72,876	22,103,741	.137	117,404	27,986,383	1,600	1,623	22,482	14.05

Year.	Ceramics.	Emery Grains.				Emery Rock.				Emery Mfrs.	Total Emery.
		Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.		
1900....	$9,143,536	661,482	300	$26,520	$0.040	11,392	11,574	$202,980	$17.82	10,006	$239,506
1901....	9,816,074	1,116,729	506	43,207	.039	12,441	12,640	240,856	19.35	10,927	294,990
1902....	9,838,426	1,665,737	756	60,079	.036	7,166	7,281	151,959	21.21	13,776	225,814
1903....	11,582,013	3,595,239	1,630	109,272	.030	10,885	11,059	188,985	17.36	23,317	321,574
1904....	11,656,686	2,281,193	1,035	109,772	.048	7,054	7,167	131,493	18.64	19,059	260,324
1905....	12,199,605	3,209,915	1,456	143,729	.045	11,073	11,250	185,689	16.77	17,996	347,414

Year.	Fertilizers.									Fuller's Earth.	
	Guano.				Crude Phosphates.				All Other.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric. Tons.	Value.	Value. per L. T.		Long Tons.	Value.
1900....	6,620	6,726	$67,413	$10.18	137,086	139,272	$791,189	$5.77	$1,400,336	....	....
1901....	4,949	5,028	71,140	14.37	175,765	178,577	872,503	4.97	1,506,965	....	....
1902....	8,407	8,542	164,783	19.60	137,386	139,584	646,264	4.70	1,725,333	....	....
1903....	21,007	21,343	251,966	12.00	132,965	134,092	679,112	5.11	2,353,496	15,267	$120,671
1904....	35,876	36,430	478,388	13.33	130,214	132,297	745,744	5.73	2,856,141	9,126	78,006
1905....	25,651	26,061	365,823	14.26	56,021	56,917	273,289	4.88	4,051,003	13,001	105,997

Year.	Gold.		Iron Ore.				Pig Iron.			
	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	$45,703,256	$21,045,828	879,831	893,908	$1,303,196	$1.48	52,565	53,406	$1,907,361	$36.28
1901...	33,237,629	21,524,251	966,950	982,421	1,659,273	1.72	62,930	63,937	1,792,014	28.48
1902...	22,710,957	21,482,360	1,165,470	1,184,118	2,583,077	2.22	619,354	629,264	10,935,831	17.66
1903..	44,054,902	21,212,794	980,440	996,127	2,261,008	2.31	599,574	609,167	11,173,302	18.64
1904..	75,646,128	9,157,106	487,613	495,415	1,101,384	2.26	79,590	80,772	1,765,107	22.20
1905...	38,557,487	11,729,077	845,651	859,181	2,062,161	2.44	212,465	215,864	5,185,784	24.41

Year.	Iron Mfrs.	Scrap, Iron and Steel.				Bar Iron.			
		Long Tons,	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.....	$8,747,089	34,431	34,982	$663,231	$19.26	19,685	19,094	$1,058,761	$56.34
1901.....	8,356,646	20,130	20,452	339,827	16.88	20,792	21,126	1,093,736	52.60
1902.....	11,779,530	109,510	111,262	1,606,720	14.67	28,844	29,307	1,286,238	44.58
1903.....	12,041,771	82,921	84,248	1,273,941	15.36	43,392	44,090	1,904,469	43.89
1904.....	10,352,931	13,461	13,676	189,506	14.08	20,905	21,247	917,254	43.88
1905.....	10,855,187	23,731	24,111	370,328	15.61	37,294	37,891	1,522,434	40.82

Year.	Rails.				Hoop, Band or Scroll.				Ingots, Blooms, Slabs, Billets, etc.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900	1,448	1,471	$56,129	$38.77	165	167	$12,409	$75.26	12,709	12,913	$1,332,896	$104.84
1901	1,905	1,935	67,052	35.19	2,974	3,021	116,841	39.29	8,164	8,295	2,340,112	164.15
1902	63,522	64,538	1,576,679	24.82	3,362	3,416	131,052	38.97	289,318	293,965	7,943,818	27.76
1903	95,555	97,083	2,159,273	22.59	1,525	1,550	74,898	49.11	261,570	265,932	7,331,299	28.03
1904	37,776	38,380	808,775	21.41	2,135	2,169	60,934	28.54	55,450	56,337	1,537,531	27.70
1905	17,278	17,554	409,807	23.72	4,772	4,848	137,612	28.84	59,285	60,234	2,072,606	34.96

Year.	Sheet, Plate and Taggers Iron or Steel.				Tin Plates, Terne Plates and Taggers Tin.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	5,143	5,226	$426,541	$82.93	60,386	61,356	$4,617,813	$76.46
1901	5,626	5,716	443,880	79.10	77,395	78,638	5,294,789	68.41
1902	7,156	7,270	545,739	76.26	60,115	61,080	4,023,421	66.93
1903	11,557	11,741	540,272	46.75	47,360	48,118	2,999,252	63.33
1904	4,165	4,232	302,500	72.63	70.652	71,782	4,354,761	61.63
1905	2,236	2,272	242,955	108.66	65,740	66,792	4.090,523	62.22

Year.	Wire-Rods.				Wire and Articles Made from.				Total Iron Imports. (e)
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons	Value.	Value per L. T.	
1900	21,092	21,430	$1,212,594	$57.49	1,848	1,877	$409,087	$221.37	$20,443,911
1901	16,804	17,073	964,744	57.40	4,129	4,192	585,354	141.77	20,404,122
1902	21,382	21,725	1,033,074	48.31	3,469	3,525	606,724	174.90	41,468,826
1903	20,836	21,169	1,028,977	49.39	5,018	5,098	728,430	145.16	41,258,864
1904	15,313	15,558	707,779	46.22	3,956	4,019	624,892	157.96	21,621,970
1905	17,616	17,898	800,027	45.41	3,978	4,042	705,465	177.34	26,392,728

Year.	Lead in Ore and Base Bullion.				Lead in Pigs and Old.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Short Tons.	Metric Tons.	Value.	Value per Sh. T.
1900	(g)114,397	103,780	$3,975,695	$34.60	....	....	....	....
1901	111,867	101,486	4,807,762	47.37	604	548	$33,882	$56.10
1902	105,186	95,425	4,424,511	46.37	2,529	2,294	132,500	52.40
1903	103,384	93,790	3,596,635	38.35	3,023	2,742	164,528	54.42
1904	104,127	94,464	3,517,691	37.24	8,724	7,914	461,316	52.88
1905	92,657	84,081	3,565,282	38.48	5,720	5,812	367,106	64.18

Year.	Lead, Sheet, Pipe, Shot, Etc.				Other Lead Mfrs.	Total Lead.	White Lead.			
	Lb.	Metric Tons.	Value.	Value per Lb.			Lb.	Metric Tons.	Value.	Value per Lb.
1900	27,945	13	$1,393	$0.050	$5,854	$3,964,942	456,872	207	$28,366	$0.062
1901	56,735	26	2,773	.048	4,654	4,849,071	384,673	174	21,226	.056
1902	224,208	102	7,765	.034	18,918	4,533,694	506,423	230	25,320	.050
1903	17,008	8	810	.048	8,071	3,770,044	453,284	206	24,595	.054
1904	69,581	32	2,441	.035	7,755	3,989,203	587,338	266	33,788	.058
1905	54,779	25	2,638	.048	4,580	3,939,606	597,510	271	34,722	.058

Yr.	Litharge.				Red Lead.				Orange Mineral.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	77,314	35	$2,852	$0.032	549,551	249	$25,532	$0.046	1,068,793	485	$61,885	$0.059
1901	49,306	22	1,873	.038	485,467	220	19,370	.040	977,644	443	52,409	.053
1902	88,115	40	2,908	.033	1,075,839	488	37,383	.035	997,494	452	49,060	.049
1903	42,756	19	1,464	.034	1,152,715	523	40,846	.035	756,742	343	36,407	.048
1904	44,541	20	1,500	.034	836,077	379	30,115	.036	766,469	348	37,178	.049
1905	117,757	53	4,139	.035	704,402	320	26,553	.038	628,003	285	31,106	.049

Year.	Magnesite.				Manganese Ore.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value	Value per L. T.
1900	....	....	....	....	256,252	260,352	$2,042,361	$7.97
1901	30,350	30,835	....	....	165,720	168,372	1,486,573	8.97
1902	45,157	45,880	$373,928	8.28	235,576	239,345	1,931,282	8.20
1903	49,684	50,479	461,399	9.29	146,056	148,393	1,278,108	8.75
1904	35,106	35,668	286,828	8.17	108,519	110,255	901,592	8.31
1905	66,405	67,566	638,619	9.46	257,033	261,146	1,952,407	7.60

Year.	Marble and Stone Mnfrs.		Metal Composition.		Mica.	Nickel. (h)	Nickel Ore and Matte.			
	Marble.	Stone. (d)	Bronze.	All Other.			Long Tons.	Metric Tons.	Value.	Value per L. T.
1900..	$945,705	$256,624	$791,306	$5,420,483	$319,560	(f)	(f)	.....	....	....
1901..	1,226,524	237,191	945,702	5,162,392	335,054	....	(f)	.....	....	....
1902..	1,435,457	222,435	816,668	5,942,453	466,332	....	(f)	.....	....	....
1903..	1,502,111	258,968	790,639	6,402,215	317,969	$207,954	11,936	16,191	$1,285,935	$80.70
1904..	1,270,443	294,035	746,073	5,909,338	269,808	206,021	8,549	8,685	915,470	107.10
1905..	1,344,525	297,154	839,133	7,152,302	403,755	335,211	13,451	13,666	1,626,920	120.95

Year.	Oil, Mineral.				Paints and Colors.	Platinum, Unmanufactured.				Platinum, Mnfrs.
	Gal.	Liters.	Value.	Value per Gal.		Lb. Troy.	Kg.	Value.	Value per Lb. Troy.	
1900...	3,039,094	11,503,913	$274,766	$0.091	$1,491,902	9,246	3,450	$1,728,777	$187.00	$36,714
1901...	2,294,684	8,686,389	151,913	.066	1,524,125	7,496	2,797	1,673,713	223.30	24,482
1902...	3,578,393	13,545,646	207,310	.058	1,745,989	8,670	3,235	1,950,362	224.96	37,618
1903...	4,266,974	16,152,203	261,199	.061	1,811,902	9,540	3,561	1,921,772	201.44	1,727,830
1904...	4,846,681	18,344,650	277,399	.057	1,526,072	8,648	3,230	1,812,242	209.55	105,636
1905...	13,725,720	52,020,479	672,127	.049	1,570,839	8,681	3,240	1,985,107	228.67	188,156

Year	Potassium Salts.											
	Chlorate.			Chloride.			Chromate and Bichromate.			Nitrate.		
	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value. per Lb.	Lb.	Value.	Value per Lb.
		$	$		$	$		$	$		$	$
1900	1,243,612	68,772	0.055	130,175,481	1,976,604	0.015	111,761	7,758	0.069	10,545,392	276,664	0.026
1901	811,127	61,348	.076	148,189,337	2,316,577	.015	430,996	29,224	.068	9 656,393	253,286	.026
1902	1,209,148	60,429	.050	140,980,460	2,141,553	.015	231,009	15,161	.066	10,505,474	299,416	.028
1903	468,042	19,308	.041	169,337,673	2,550,478	.015	41,229	2,784	.067	13,835,668	367,721	.026
1904	95,889	4,209	.044	174,865,872	2,832,554	.016	26,053	1,817	.069	14,184,287	376,931	.027
1905	42,510	2,876	.067	214,207,064	3,326,748	.016	59,650	4,225	.070	9,911,534	304,596	.027

Year.	Potassium Salts. All Other.			Precious Stones.			Pyrites (i).			
	Lb.	Value.	Value per Lb.	Uncut.	Cut, not Set.	Jewelry.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	54,904,088	$1,407,303	$0.025	$3,751,219	$9,612,127	(f)	332,517	337,837	$1,095,598	$3.30
1901...	72,489,913	1,636,856	.022	6,637,860	17,166,049	(f)	398,969	405,353	1,407,244	3.53
1902...	91,857,009	1,820,585	.020	8,282,760	18,494,288	(f)	437,319	444.316	1,623,430	3.71
1903...	70,205,850	1,593,380	.023	10,374,877	15,428,819	$954,456	427,319	434,156	1,636,450	3.83
1904...	74,720,241	1,678,699	.023	10,386,341	15,254,293	803,952	413,585	420,202	1,533,564	3.73
1905...	82,935,632	1,891,081	.023	10,401,514	24,406,183	801,566	515,722	520,926	1,780,800	3.47

Year.	Salt.				Silver.		Sodium Nitrate.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	207,933	188,636	$633,192	$3.05	$14,695,965	$25,404,378	182,108	185,022	$4,935,520	$27.12
1901	194,967	176,872	670,648	3.44	12,957,987	18,188,795	208,654	211,992	5,997,595	28.82
1902	188,775	167,481	654,990	3.47	8,502,614	17.900,321	205,245	208,529	5,996,205	29.21
1903	157,201	142,494	489,179	3.11	7,935,844	16,038,664	272,947	277,314	8,700,806	31.88
1904	167,295	151,810	515,822	3.08	11,865,805	14,221,237	228,012	231,660	9,333,613	32.41
1905	158,449	143,783	491,079	3.10	16,477,701	19,465,242	321,231	326,371	11,206,548	34.89

Year.	Sodium Hydroxide (Caustic).			Soda Ash and Carbonate.			All Other Sodium Salts.		
	Lb.	Value	Value. per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.
1900....	8,403,749	$150,530	$0.018	73,815,425	$613,379	$0.008	20,484,938	$259,802	$0.012
1901....	3,812,847	94,303	.025	31,415,788	276,261	.009	14,491,559	189,543	.013
1902....	3,334,697	77,482	.020	31,889,252	284,634	.009	17,151,682	283,745	.016
1903....	2,970,426	73,647	.025	25,313,370	228,041	.009	14,272,646	268,738	.019
1904....	2,570,984	64,405	.025	23,631,832	205,496	.009	10,399,711	281,527	.027
1905....	2,245,789	56,515	.025	15,754,979	146,812	.009	11,257,629	247,413	.022

Year.	Sulphur.											
	Crude.				Flowers.				Refined.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900.........	166,457	169,120	$2,918,610	$17.53	628	638	$17,437	$27.77	243	247	$6,279	$25.84
1901.........	174,162	176,949	3,256,951	18.70	748	761	20,201	26.98	268	272	6,308	23.57
1902.........	176,951	179,782	3,360,562	19.00	738	750	19,954	27.04	14	15	369	24.99
1903.........	188,888	191,910	1,649,756	19.32	1,854	1,883	52,680	28.42	189	192	7,254	38.44
1904.........	128,885	130,947	2,463,779	19.12	1,332	1,353	39,133	29.38	204	207	9,776	47.92
1905.........	83,201	84,532	1,522,005	18.29	572	581	16,037	28.04	778	790	19,960	25.66

Year.	Talc.				Tin.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900..................	79	72	$1,070	$13.54	69,989,502	31,747	$19,458,586	$0.278
1901..................	2,386	2,164	27,015	11.74	74,560,487	33,820	19,024,761	.255
1902..................	2,859	2,594	35,336	12.35	85,043,353	38,575	21,263,337	.250
1903..................	1,790	1,623	19,635	11.00	83,133,847	37,702	22,265,367	.268
1904..................	3,268	2,964	36,370	11.13	83,168,657	37,718	22,356,896	.270
1905..................	4,000	3,630	48,225	12.06	89,227,698	40,507	26,316,023	.294

Year.	Zinc.								
	Blocks, Pigs and Old.				Oxide (i).		Sulphide.		Mfrs.
	Lb.	Metric Tons.	Value	Value. per Lb.	Lb.	Value.	Lb.	Value.	Value.
1900......	2,013,196	913	$97,772	$0.048	2,657,514	....	....	....	$36,836
1901......	775,881	352	30,920	.040	3,327,976	....	....	....	42,643
1902......	1,238,091	561	46,713	.038	3,434,466	....	1,247,936	$32,879	37,191
1903......	728,614	330	30,900	.042	3,653,076	$188,495	1,229,806	33,077	18,938
1904......	933,474	423	44,455	.048	2,809,905	165,110	1,228,875	31,382	11,918
1905......	1,042,081	473	51,052	.048	3,779,311	196,220	1,235,360	33,308	12,390

(a) From Summary of Commerce and Finance of the United States. (b) Includes arsenic sulphide. (c) Barrels of 400 lb. (d) Including slate. (e) Not including iron ore. (f) Not reported. (g) Includes pig and old. (h) Includes nickel oxide, alloys in which nickel is the principal constituent and manufactures of nickel. (i) Containing more than 25% sulphur. (j) Includes white pigments containing zinc but not lead, dry and in oil.

EXPORTS OF DOMESTIC PRODUCTS. (*a*)

Year.	Aluminum and Mfrs. of.	Asbestos and Mfrs. of.	Brass and Mfrs. of.	Brick.	Cement.				Chemicals and Drugs.
					Bbl. (*i*)	Metric Tons.	Value.	Value per Bbl.	
1900.....	$281,821	$124,971	$2,063,072	....	100,400	18,216	225,306	$2.24	$13,771,682
1901.....	183,579	113,316	2,078,178	$541,589	373,934	67,393	679,296	1.82	14,267,110
1902.....	116,052	130,437	1,809,312	501,434	340,821	61,838	526,471	1.54	13,437,367
1903.....	157,187	158,360	2,063,569	439,277	285,463	51,748	433,984	1.52	14,276,465
1904.....	166,876	223,096	3,093,803	587,385	774,940	140,898	1,104,086	1.42	14,821,808
1905.....	290,777	....	3,055,189	799,878	1,026,502	186,298	1,387,906	1.35	17,229,084

Year.	Coal. Anthracite.				Coal. Bituminous.				Coke.	
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Value.
1900.	1,654,610	1,681,084	$7,092,489	$4.29	6,262,909	2,363,631	$14,431,590	$2.31	376,999	$1,358,968
1901.	1,993,307	2,025,200	8,937,147	$4.48	5,390,086	3,476,327	13,085,763	2.53	384,330	1,516,898
1902.	907,977	922,505	4,301,946	4.73	5,218,969	5‘302,472	13,927,063	2.66	392,491	1,785,188
1903.	2,008,857	2,040,999	9,780,044	4.86	6,303,241	6,404,093	17,410,385	2.76	416,385	2,091,875
1904.	2,228,392	2,264,046	11,077,570	4.97	6,345,126	6,446,648	17,160,538	2.74	523,090	2,311,401
1905.	2,229,983	2,265,663	11,104,654	4.98	6,959,265	7,070,613	17,867,964	2.56	599,054	2,243,010

Year.	Copper. In Ore and Matte (*b*).				Copper. Ingots, Bars, Plates and Old.				Copper. Mfrs.	Copper. Total Except Ore.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.	Value.	
1900...	10,007	10,168	$1,332,829	$133.18	337,973,751	153,304	$55,285,047	$0.164	$2,257,563	$57,542,610
1901...	19,613	19,924	2,536,549	129.40	194,249,828	88,111	31,692,563	.164	1,842,336	33,384,899
1902...	18,035	18,321	1,326,131	73.53	354,668,849	160,876	43,392,800	.122	2,092,798	45,485,598
1903...	12,291	12,488	855,367	69.59	310,729,524	140,920	41,170,059	.132	2,339,729	43,509,788
1904...	18,927	19,230	1,202,537	63.54	554,550,030	251,497	71,488,116	.129	3,328,818	74,816,934
1905...	37,688	38,291	1,531,429	40.63	534,907,619	242,699	80,693,232	.151	4,184,070	84,877,302

Year.	Earthen and China Ware	Fertilizers. Crude Phosphates.				Fertilizers. All Other.				Glassware	Gold. In Coin and Bullion (*c*)	Gold. In Ore. (*d*)
		Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.			
1900	$558,794	619,995	629,915	$5,217,560	$8.38	25,976	26,392	$537,908	$20.71	$2,042,633	$54,064,697	$69,926
1901	526,820	729,539	741,212	5,839,245	8.01	14,153	14,379	332,964	23.54	2,087,043	56,717,350	1,012,589
1902	604,646	802,086	814,919	6,193,372	7.73	16,451	16,714	383,438	23.31	2,094,701	35,722,835	307,756
1903	589,001	785,259	797,823	6,109,230	7.78	20,343	20,668	557,059	27.38	2,053,516	43,765,360	581,474
1904	791,739	842,484	855,964	6,521,555	7.74	25,549	25,958	714,367	27.96	2,130,297	120,226,424	985,403
1905	983,554	934,940	949,899	7,465,592	7.91	22,865	23,231	732,744	32.05	2,252 399	46,099,580	694,887

Year.	Gunpowder and Other Explosives.	Iron. Ore. Long Tons.	Iron. Ore. Metric Tons.	Iron. Ore. Value.	Iron. Ore. Value per L. T.	Iron. Pig. Long Tons.	Iron. Pig. Metric Tons.	Iron. Pig. Value.	Iron. Pig. Value Per L. T.
1900	....	51,460	52,283	$154,756	$3.01	286,687	291,404	$4,654,582	$16.23
1901	$1,965,875	64,703	65,748	163,465	2.54	81,211	82,510	1,257,699	15.65
1902	2,393,480	88,445	89,860	294,168	3.32	27,487	27,927	502,947	18.30
1903	2,367,148	80,611	81,901	255,728	3.17	20,379	20,705	384,334	18.86
1904	2,466,278	213,865	217,287	458,823	2.14	49,025	49,809	764,543	15.60
1905	2,848,155	208,058	211,387	532,457	2.56	49,221	50,009	762,899	15.50

Year.	Iron, Bar. Long Tons.	Iron, Bar. Metric Tons.	Iron, Bar. Value.	Iron, Bar. Value per L. T.	Iron, Band, Hoop and Scroll. Long Tons.	Iron, Band, Hoop and Scroll. Metric Tons.	Iron, Band, Hoop and Scroll. Value.	Iron, Band, Hoop and Scroll. Value per L. T.	Billets, Ingots and Blooms. Long Tons.	Billets, Ingots and Blooms. Metric Tons.	Billets, Ingots and Blooms. Value.	Billets, Ingots and Blooms. Value per L. T.
1900	13,298	13,512	$558,576	$42.04	2,976	3,024	$137,437	$46.20	107,385	109,103	$2,915,371	$27.15
1901	17,708	17,993	674,671	38.16	1,561	1,586	74,056	47.44	28,614	29,072	708,887	24.78
1902	22,249	22,605	869,519	39.08	1,674	1,701	82,322	49.18	2,409	2,447	74,938	31.11
1903	19,380	19,690	796,631	41.11	1,241	2,175	101,839	47.56	5,445	5 532	141,924	26.07
1904	29,582	30,055	1,133,128	34.93	3,435	3,489	162,039	47.18	314,324	319,353	6,150,035	19.56
1905	32,025	32,537	1,255,418	39.20	4,426	4,497	182,431	41.22	237,638	241,440	4,701,909	19.79

Year.	Iron, Nails and Spikes, Cut. Lb.	Iron, Nails and Spikes, Cut. Metric Tons.	Iron, Nails and Spikes, Cut. Value.	Iron, Nails and Spikes, Cut. Value per Lb.	Iron, Nails and Spikes, All Other. Lb.	Iron, Nails and Spikes, All Other. Metric Tons.	Iron, Nails and Spikes, All Other. Value.	Iron, Nails and Spikes, All Other. Value per Lb.	Iron, Plates and Sheets. Long Tons.	Iron, Plates and Sheets. Met. Tons.	Iron, Plates and Sheets. Value.	Iron, Plates and Sheets. Value per L. T.
1900	25,005,308	11,342	$626,497	$0.025	65,444,387	29,681	$1,816,813	$0.028	9,331	9,481	$600,600	$64.35
1901	20,835,944	9,452	450,331	.021	46,298,262	21,001	1,152,368	.025	6,909	7,020	452,695	65.52
1902	16,122,775	7,312	339,227	.021	64,565,650	29,287	1,456,768	.022	3,434	3,489	229,887	66.94
1903	19,912,563	9,031	424,985	.021	75,654,532	34,310	1,698,500	.024	4,782	4,858	273,618	57.22
1904	20,772,049	9,422	416,389	.020	80,279,746	36 403	1,949,908	.024	4,728	4,804	247,694	52.39
1905	17,674,099	8,019	352,405	.020	89,276,058	40,506	2,108,836	.024	8,004	8,132	460,995	57.60

Year.	Steel, Sheets and Plates. Long Tons.	Steel, Sheets and Plates. Metric Tons.	Steel, Sheets and Plates. Value.	Steel, Sheets and Plates. Value per L. T.	Iron Rails. Long Tons.	Iron Rails. Metric Tons.	Iron Rails. Value.	Iron Rails. Value per L. T.	Steel Rails. Long Tons.	Steel Rails. Metric Tons.	Steel Rails. Value.	Steel Rails. Value per L. T.
1900.	45,534	46,264	$1,638,478	$35.98	5,374	5,460	$119,206	$22.18	356,445	361,945	$10,895,416	$30.58
1901.	23,923	24,303	959,471	40.11	901	915	32,357	35.93	318,055	323,044	8,628,781	27.14
1902.	14,866	15,104	725,547	48.80	211	214	4,639	22.02	67,455	68,534	1,902,396	28.09
1903.	13,312	13,525	657,713	49.47	181	184	8,808	48.67	30,656	31,146	937,779	30.59
1904.	50,477	51,278	2,064,241	40.89	1,405	1,427	23,870	17.00	414,845	421,482	10,661,222	25.72
1905.	67,093	68,166	2,389,084	43.06	*Nil.*	....	....	....	295,023	299,473	7,310,029	24.78

Ye	Structural Iron and Steel. Long Tons.	Structural Iron and Steel. Metric Tons.	Structural Iron and Steel. Value.	Structural Iron and Steel. Value per L. T.	Wire. Long Tons.	Wire. Metric Tons.	Wire. Value.	Wire. Value per L T.	Steel Wire Rods. Long Tons.	Steel Wire Rods. Metric Tons.	Steel Wire Rods. Value.	Steel Wire Rods. Value per L. T.
1900.	67,714	68,797	$3,570,769	$52.73	78,014	79,262	$4,604,047	$59.77	10,652	10,822	$505,529	$47.3[illegible]
1901.	54,005	54,869	3,031,861	56.10	88,238	89,650	4,805,608	54.36	8,165	8,296	271,552	33.2[illegible]
1902.	53,859	54,721	2,828,460	52.52	97,843	99,414	5,140,702	52.54	24,613	25,007	831,067	33.7[illegible]
1903.	30,641	31,131	1,788,556	58.37	108,521	110,258	5,528,726	50.94	22,360	22,718	713,718	31.9[illegible]
1904.	55,514	56,402	2,777,768	50.04	118,581	120,478	5,935,093	50.05	20,073	20.394	695,448	34.64
1905.	83,193	84,524	4,355,186	52.35	142,601	144,883	7,061,442	49.52	6,514	6,618	277,651	42.6[illegible]

Year.	Lead and Mfrs. of.	Marble, Stone.	Mica.	Nickel. (e)	Petroleum products. (In Thousands of Units.)* Crude. M Gals.	M Liters.	M Value.	Value per Gal.	Naphtha. M Gals.	M Liters.	M Value.	Value per Gal.
1900	$459,574	$1,556,981	$165	$1,382,727	138,161	523,000	$7,341	$0.053	18,570	70,295	$1,681	$0.081
1901	625,234	1,785,515	3,584	1,521,291	127,008	480,781	6,038	.050	21,685	82,087	1,742	.079
1902	696,010	1,587,957	....	924,579	145 234	549,775	6,331	.042	19,683	74,509	1,393	.071
1903	491,362	1,688,316	760	703,550	126,512	478,847	6,782	.054	12,973	49,103	1,519	.017
1904	616,126	1,337,754	1,770	2,130,933	111,176	420,801	6,351	.057	24,989	94,583	2,322	.093
1905	667,861	1,384,208	....	2,894,700	126,185	447,610	6,086	.048	28,420	107,570	2,215	.078

Year.	Petroleum Products. (In Thousands of Units.)* Illuminating Oil. M Gals.	M Liters.	M Value.	Value per Gal.	Lubricating Oil. M Gals.	M Liters.	M Value.	Value per Gal.	Residue, Etc. (g) M Gals.	M Liters.	M Value.	Value per Gal.
1900...	739,163	3,104,593	$54,693	$0.074	71,211	269,540	$9,933	$0.139	19,750	74,760	$845	$0.042
1901...	827,479	3,131,399	53,491	.065	75,306	285,010	10,260	.136	27,596	104,463	1,255	.046
1902...	778,801	2,947,762	49,079	.063	82,201	311,163	10,872	.133	38,316	145,043	922	.024
1903...	691,837	2,618,603	51,356	.074	95,622	361,929	12,690	.133	9,753	36,916	282	.029
1904...	761,358	2,881,740	58,384	.077	89,738	339,469	12,389	.138	34,904	132,072	1,174	.034
1905...	881,450	3,336,288	54,901	.062	113,730	430,468	14,312	.126	70,728	267,705	2,128	.030

Year.	Petroleum Products.* Paraffin. M Lb.	M Metric Tons.	M Value.	Value per Lb.	Quicksilver. Lb.	Metric Tons.	Value.	Value per Lb.	Silver. In Coin and Bullion (c)	In Ore (d)
1900........	157,108	71.2	$8,186	$0.052	778,191	353	$425,812	$0.547	$65,705,909	$515,755
1901........	151,694	68.8	7,960	.052	843,938	383	475,609	.563	55,526,975	111,383
1902........	175,269	79.5	8,398	.048	1,013,434	459	575,099	.568	49,228,303	44,651
1903........	204,120	92.6	9,596	.047	1,344,615	610	719,119	.535	4,0531,095	79,247
1904........	174,582	79.2	8,273	.047	1,611,365	731	847,108	.526	49,975,370	159,875
1905........	160,836	73.0	7,873	.049	1,009,446	458	497,470	.493	52,300,698	5,212,404

Year.	Tin Mfrs.	Zinc Ore. Long Tons.	Metric Tons.	Value.	Value per L. T.	Zinc Pigs, Bars, Plates and Sheet. Lb.	Metric Tons.	Value.	Value per Lb.
1900............	$467,354	37,555	38,158	$1,133,633	$30.19	44,802,577	20,322	$2,217,693	$0.050
1901............	495,435	39,425	40,056	1,167,684	29.62	6,780,221	3,071	288,906	.043
1902............	529,061	49,762	50,558	1,449,104	29.12	6,473,135	2,936	300,557	.046
1903............	777,917	35,188	35,751	987,000	28.05	3,041,911	1,380	163,379	.053
1904............	701,625	32,063	32,576	905,782	28.25	20,145,942	9,204	1,094,490	.053
1905............	930,844	27,630	28,072	848,451	30.71	11,031,812	5,005	682,254	.062

Year.	Zinc Oxide. Lb.	Metric Tons.	Value.	Value per Lb.	Zinc Mfrs.	Total (Except Ore).
1900..............	11,391,666	5,167	$496,380	$.044	$99,288	$2,813,361
1901..............	9,122,283	4,138	393,259	.043	82,046	764,211
1902..............	10,716,364	4,861	433,722	.040	114,197	848,476
1903..............	14,429,885	6 544	578,215	.041	71,354	812,948
1904..............	16,313,826	7,399	628,494	.039	117,957	1,840,941
1905..............	22,560,625	10,236	810,203	.036	159,995	1,652,452

RE-EXPORTS OF FOREIGN PRODUCTS. (*a*)

Year.	Antimony.				Antimony Ore.			
	Lb.	Metric Tons.	Value.	Value per lb.	Short Tons.	Metric Tons.	Value.	Value per Sh. T.
1900	23,520	10.7	$2,352	$0.100	*Nil.*	....	....	....
1901	*Nil.*	....	....	....	25	22.1	$1,536	$63.05
1902	37,184	16.9	2,710	.073	104	94.6	4,602	44.13
1903	79,917	36.0	4,478	.056	*Nil.*	....	....	....
1904	31,077	14.0	1,734	.056	214	194.0	10,775	50.35
1905	*Nil.*	....	....	....	....	....	....	....

Year.	Asphaltum, Crude.				Brass and Mfrs.	Cement.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.		Bbl. (*i*)	Metric Tons.	Value.	Value per Bbl.
1900	629	639	$10,044	$15,98	$2,155	39,540	7,174	$63,880	$1.62
1901	2,209	2,244	18,078	8,19	813	43,691	7,927	72,761	1.67
1902	2,930	2,977	23,564	8,11	938	32,594	5,913	48,797	1.50
1903	1,605	1,631	13,894	8,66	7,576	25,362	4,601	32,156	1.27
1904	1,887	1,917	26,272	13,92	2,517	39,711	7,186	54,486	1.37
1905	1,081	1,098	18,190	16,83	1,228	31,874	32,384	40,583	1.27

Year.	Chemicals.											
	Salts of Potassium. (*f*)				Chloride of Lime.				Nitrate of Sodium.			
	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900	808,701	366,824	$43,524	$0.054	148,116	67,185	$1,987	$0.014	3,089	3,139	$112,550	$36.43
1901	633,100	287,182	43,446	.068	13.916	6,312	312	.023	2,482	2,519	101,489	40.90
1902	1,266,145	574,323	59,789	.048	198,794	90,172	2,997	.014	3,675	3,734	144,650	39.36
1903	1,299,905	589,637	33,264	.026	836.411	379,696	7,609	.009	4,417	4,488	184,657	41.81
1904	1,262,222	572,544	33,358	.027	1.434	650	13	.009	6,076	6,173	279,864	46.06
1905	3,053,191	1,386,149	83,652	.027	100	102	3	.003	8,991	9,135	420,613	46.78

Year.	Chemicals. (*Continued.*)											
	Caustic Soda.				Soda Ash and Carbonate.				Sodium Salts, All Other.			
	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.
1900	1,139,954	517,080	$24,228	$0.023	78,017	35,388	$1,126	$0.014	270,307	122,610	$2,788	$0.009
1901	1,001,940	452,482	21,511	.023	369,521	167,614	5,184	.014	133,400	60,510	3,398	.023
1902	1,343,132	609,246	28,704	.023	62,653	28,419	931	.014	115,491	52,386	1,626	.014
1903	1,116,354	506,378	23,227	.028	30,030	13,622	464	.015	42,540	19,294	437	.010
1904	1,115,600	506,036	23,608	.021	40.351	18,303	593	.014	1,778,616	806,780	25,312	.014
1905	1,087,772	493,848	22,728	.021	32.221	14,628	473	.015	16,748	7,604	177	.011

Year.	Clay or Earths.				Coal, Bituminous.				Copper. Ore and Matte.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900	78	79	$572	$7.34	6,740	6,848	$19,740	$2.93	964	979	$170,191	$176.53
1901	80	81	825	10.34	3,796	4,403	10,627	2.45	9,891	10,050	1,406,648	142.19
1902	123	125	1,284	10.43	7,559	7,680	22,153	2.93	14,446	14,657	2,229,912	154.57
1903	88	89	621	7.06	88,468	89,883	453,613	5.13	5,750	5,232	852,726	165.58
1904	210	214	2,466	11.74	7,250	7,366	21.910	3.02	*Nil.*	......	....	....
1905	102	104	954	9.35	3,945	4,008	10,974	2.78	*Nil.*	......	....	....

Year.	Copper. (*Continued.*) Pigs, Bars, Ingots, Old and All Unmanufactured. Lb.	Metric Tons.	Value.	Value per Lb.	Mfrs.	Earthen, Stone and China Ware	Fertilizers. Total Value.	Glass and Glassware.	Graphite. Long Tons.	Value.
1900...	1,281,782	581	$212,264	$0.166	$21,032	$38,008	$32,102	$14,614	3	$115
1901...	12,888,083	5,846	2,145,468	.166	9,462	24,080	2,833	16,749	*Nil.*	....
1902...	11,629,877	5,275	1,604,522	.138	10,939	18,989	31,476	34,236	12	834
1903...	2,093,103	949	261,413	.125	13,027	19,411	3,281	19,116	63	4,223
1904...	1,088,672	494	140,695	.129	19,461	32,640	139,363	20,522	8	455
1905...	1,718,584	780	272,945	.159	12,621	30,455	8,984	34,552	5	91

Year.	Iron and Steel. Pig Iron. Long Tons.	Metric Tons.	Value.	Value per L. T.	Scrap. Long Tons.	Metric Tons.	Value.	Value per L. T.	Bar Iron. Long Tons.	Metric Tons.	Value.	Value per L. T.
1900....	151	153	$6,579	$43.69	9,079	9,224	$131,241	$14.46	48	49	$2,447	$51.00
1901....	189	191	6,148	32.70	3,331	3,384	51.663	18.51	67	68	7,569	113.00
1902....	250	254	6,286	25.14	1,542	1,567	25,020	*a* 16.23	22	22	1,875	86.56
1903....	1,863	1,893	33,996	18.25	262	266	2,862	10.92	16	16	2,108	130.93
1904....	1,646	1,672	25,910	15.74	190	193	2,367	12.46	7	7	765	102.55
1905....	1,010	1,026	29,047	28.76	4,270	4,338	80,623	18.88	22	22	2,556	118.18

Year.	Iron and Steel. (*Continued.*) Rails. Long Tons.	Metric Tons.	Value.	Value per L. T.	Steel, Ingots, Blooms, Etc. Long Tons.	Metric Tons.	Value.	Value per L. T.	Sheets, Plates, Rods, Wire. Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900....	*Nil.*	....	....	....	2	2	$1,342	$559.33	209	213	11,599	$55.41
1901....	*Nil.*	....	....	....	2	2	1,059	504.30	190	193	17,272	90.81
1902....	297	302	$7,184	$24.20	106	108	6,774	64.09	236	240	14,221	60.26
1903....	739	751	17,560	23.76	60	61	5,316	88.01	55	56	5,532	100.04
1904 ..	96	98	2,305	24.00	40	41	6,208	154.81	108	110	6,482	60.01
1905....	31	31	1,132	36.52	86	87	15,570	181.05	161	164	8,019	49.81

Year.	Iron and Steel. (*Concluded.*) Tin and Terne Plates, Taggers Tin. Long Tons.	Metric Tons.	Value.	Value per L. T.	Mfrs.	Lead and Mfrs.	Marble and Stone Mfrs. (*h*)	Metal Composition
1900..	464	470	$37,395	$80.60	$328,704	$3,843,881	$5,721	$79,218
1901..	118	120	8,519	72.20	149,771	4,190,525	17,063	35,438
1902..	98	100	7,471	76.24	242,225	3,553,144	11,210	108,575
1903..	2	2	184	118.71	399,147	2,917,957	7,389	54,490
1904..	81	82	5,306	65.86	518,564	2,880,907	21,055	42,730
1905..	26	26	3,014	115.92	544,321	2,441,166	13,653	90,999

Year.	Paints and Colors.	Salt.				Sulphur—Crude.			
		Lb.	Metric Tons.	Value	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.....	$13,814	3,548,724	1,610	$3,907	$0.0011	590	599	$13,495	$22.89
1901.....	17,923	3,699,411	1,678	7,155	.0019	207	210	5,086	24.60
1902.....	14,217	2,310,759	1,048	4,544	.0020	1,253	1,273	28,024	22.37
1903.....	13,467	7,804,215	3,585	26,636	.0034	967	982	22,658	23.43
1904.....	11,888	2,089,234	948	2,814	.0013	2,493	2,533	58,887	23.62
1905.....	14,227	611,912	278	893	.0015	1,713	1,741	36,858	21.52

Year.	Tin in Blocks, Pig and Granulated.				Zinc and Mfrs.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	
1900..	495	503	$335,377	$677.96	$3,048
1901..	939	954	562,350	598.89	1,641
1902..	479	486	286,897	598.95	765
1903..	512	520	317,805	620.47	2,362
1904..	519	527	322,234	620.87	1,236
1905..	557	567	375,763	674.62	1,831

*For convenience in tabulating, the quantities of all the petroleum products and their gross values have been divided by 1,000.

(*a*) From Summary of Commerce and Finance of the United States.

(*b*) Very little ore, mainly matte.

(*c*) Total exports of coin and bullion; that is, includes both foreign and domestic.

(*d*) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte, which is classified as copper ore, and no record is kept of its silver contents. The gold in copper matte exported is not included in the exports of gold given in the above table. These figures include ore of both domestic and foreign origin.

(*e*) Includes nickel oxide and nickel matte.

(*f*) Includes chlorate, chloride, nitrate and all other salts of potassium.

(*g*) Reported in bbl., but calculated to gal., on a basis of 42 gal. to the bbl.

(*h*) Includes slate.

(*i*) Barrel of 400 lb.

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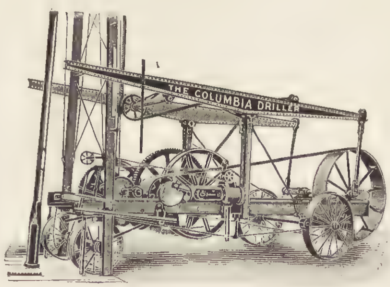

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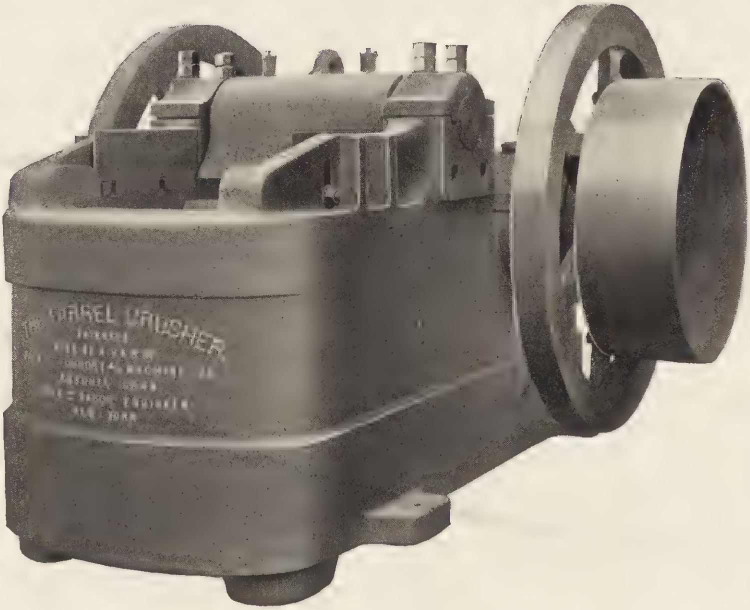

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